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Evaluation of the Hoboken Converter at Glogow, Poland

Instytut Metali Niezelaznych
Gliwice (Poland)

Prepared for

Industrial Environmental Research
Lab -Cincinnati, OH

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NTIS

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Environmental Protection
Agency

Industrial Environmental Research
Laboratory
Cincinnati OH 45268

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EPA

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16. ABSTRACT

In 1975, the U. S. Environmental Protection Agency awarded a contract to the Ministry of Smelting Poland for research to minimize emissions of fugitive pollutants from copper smelters and to assist in the control of smelter pollutants. The project objectives were to develop procedures for operating copper converters for steady gas flow containing relatively high concentrations of SO_2; to improve cleaning and treating of particulates in the converter gas streams to allow better operation of SO_2 removal systems, such as contact sulfuric acid plants; and to show how procedures and results that were developed could be applied to various types of copper smelters encountered in industry. During the course of the project, portions of the PL-480 funding were utilized to evaluate the Hoboken Converter, a potential substitute process providing stricter environmental control of fugitive SO_2 and particulate from copper converting.

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EVALUATION OF THE HOBOKEN CONVERTER AT GLOGOW, POLAND

by

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Gliwice, Poland

Contract No. 5-533-5

Project Officer

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FOREWORD

When energy and material resources are extracted, processed, converted, and used, the pollution related impacts on our environment and even on our health often require that new and increasingly more efficient pollution control methods be used. The Industrial Environmental Research Laboratory - Cincinnati assists in developing and demonstrating new and improved methodologies that will meet these needs both efficiently and economically.

This report presents the results of an investigation into the control of air pollutant emissions from the converting process in the primary copper industry. The study was performed to develop improved methods for operation of the copper converter to allow more effective control of sulfur oxides and particulate matter. The results are being used within the Agency's Office of Research and Development as part of a larger effort to develop improved technologies for reducing pollutant discharges in the nonferrous metals industries. The findings will also be useful to other Agency components and industry in dealing with environmental control problems. The Metals and Inorganic Chemicals Branch of the Industrial Pollution Control Division should be contacted for any additional information concerning this program.

David G. Stephan
Director
Industrial Environmental Research Laboratory
Cincinnati

PREFACE

In 1975, the U.S. Environmental Protection Agency (EPA) awarded a contract to the Ministry of Smelting in Poland for research to minimize the emission of fugitive pollutants from copper smelters and to assist in the control of all smelter pollutants. The Ministry of Smelting assigned this work to the Institute of Nonferrous Metals in Gliwice, which conducts both basic and applied research and development for the nonferrous metals industries in Poland. This work was accomplished by the research workers at the Institute, with the assistance and close cooperation of the staffs of the Polish copper smelters, the design bureau BP "BIPROMET," and the Association of Nonferrous Mining and Metallurgy.

This project was initiated with EPA's Office of Research and Monitoring, Control Systems Laboratory, and subsequently transferred to the Industrial Environmental Research Laboratory, Industrial Pollution Control Division, following an internal EPA reorganization. It was supported by PL-480 funds through EPA's Special Foreign Currency Branch.

ABSTRACT

This research project was initiated with three overall objectives--to develop procedures for operating copper converters to provide steady gas flows containing relatively high concentrations of SO_2; to improve cleaning and treating of particulates in converter gas streams to allow better operation of SO_2 removal systems such as contact sulfuric acid plants; and to show how the procedures and results that were developed could be applied to the various types of copper smelters encountered in industry.

The intermittent flow of gases from the converter complicates the operation of the sulfuric acid plants typically used to further process SO_2. In addition, the converter is a significant source of fugitive emissions. The Polish copper industry employs the siphon converter exclusively. The most important difference between this converter and the more commonly employed Peirce-Smith is that in the siphon converter the gas offtake is separated from the charging and pouring port. This design thus has the potential to reduce both air dilution of the gases and fugitive emissions through the converter mouth.

The Institute of Nonferrous Metals conducted project research in both the laboratory and in full-scale converters. It was demonstrated that the operation of two or more converters could be properly scheduled to ensure a steady high-strength gas stream suitable for feed to the acid plant. The gas cleaning system that achieves these results consists of gas coolers and dry electrostatic precipitators, all synchronized with the converter and acid plant operations by a two-stage gas pumping system. Some problems remain in stabilizing gas removal from the converters while simultaneously minimizing air dilution of the gases, fugitive emissions through the mouth, and carryover of materials from the bath into the gas offtake. During normal operation, however, the system effectively controls sulfur emissions, with the converters the only source of feed gas to the acid plant. With appropriate modifications, this gas cleaning system could be adapted to other smelters, including those using Peirce-Smith converters.

This report was submitted in fulfillment of Contract No. 5-533-5 by the Institute of Nonferrous Metals under the sponsorship of the U.S. Environmental Protection Agency. It covers the period from March 1975 to February 1978, and work was completed in December 1978.

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Project Officer for the Industrial Environmental Research Laboratory of the U.S. Environmental Protection Agency was George S. Thompson, Jr., with associated project input by Alfred B. Craig, Jr. Project activities were coordinated by Thomas J. LePine of EPA's Special Foreign Currency Branch.

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SECTION 1

INTRODUCTION

Copper smelters contain a variety of continuous and batch pyrometallurgical processes that are sources of emissions of sulfur oxides and particulate matter that are difficult to control. One such process that requires SO_2 control is the converter, a batch-type operation in which the copper/iron/sulfur matte formed in a smelting furnace is blown with compressed air to oxidize the iron and copper sulfides, producing crude (blister) copper and an iron silicate slag. The intermittent flow of gases from the converter complicates the operation of the sulfuric acid plants typically used to further process the SO_2. The converter is also frequently the most significant source of fugitive emissions at a copper smelter.

No method has been found to eliminate the converting process from conventional copper smelting. The most common type of converter used in U.S. copper smelters is the Peirce-Smith, a horizontal, refractory-lined, cylindrical furnace with an opening in the side that serves as a mouth for charging feed materials and pouring off molten blister copper and slag. The mouth of the Peirce-Smith also serves as the exhaust port for the off-gases, which are partially collected by a loose-fitting hood situated above the converter. The hood must be retracted during charging and pouring operations, which permits large quantities of fugitive emissions to escape and causes additional air dilution of the gas stream, thereby decreasing gas temperature and SO_2 content. A variety of modifications have been made to the design of the hood and converter mouth to minimize these emissions. Retractable gates, secondary hoods, enclosed converters, and computerized control of the gas off-take based upon blast intensity or gas pressure have been reported to be used at some U.S. smelters.

The siphon converter is an alternative design to the Peirce-Smith that is intended to eliminate the problem of excess air infiltration into the gas removal and cleaning system. The basic furnace design is similar to the Peirce-Smith. However, instead of allowing the gases to exit through the converter mouth, the siphon converter has an integral side flue located at one end. Shaped like an inverted "U," this flue, or siphon, rotates with the converter. It is attached to a cylindrical duct, which also rotates, that leads to a fixed vent flue. This design provides an airtight connection between the converter and the gas removal ductwork. It thus minimizes gas dilution by external air and facilitates the maintenance of the high-SO_2 stream desirable for sulfur recovery in a conventional sulfuric acid plant.

Siphon converters are in only limited use in the world copper industry. The first such units to be designed are still being operated at the Hoboken smelter in Belgium; larger models have been installed under license from Mechim at the Inspiration Consolidated Copper smelter in Miami, Arizona, and are reported to be used at several smelters in other countries. Siphon units are used in the three modern copper smelters in Poland. At the oldest of these, small converters with capacities of 33 Mg of copper per cycle were constructed, using furnace drawings and the gas take-off design purchased from Mechim. At the two newer smelters, Glogow Nos. 1 and 2, larger converters with capacities of 80 Mg of copper have been designed and installed. Polish engineers have incorporated a number of modifications to the design of both the converter and the gas cleaning system, and these units may represent second- and third-generation improvements of the original Hoboken design.

In order to furnish information to the United States Environmental Protection Agency on how these converters are used to minimize the emissions of fugitive pollutants from copper converters and optimize the control of furnace off-gases, the following objectives were developed for this study:

- Develop procedures for operating copper converters to provide steady gas flows containing relatively high concentrations of SO_2. The off-gases from converters are normally used for recovery of sulfuric acid by the contact method. At some smelters, these gas streams are combined with those from fluidized-bed roasters or continuous smelting furnaces. In such cases, even the leanest converter gases can be processed as they are combined with other SO_2-rich gases. In the majority of smelters, however, sulfuric acid is recovered exclusively from the converter gases. Because converting is a batch operation with a fluctuating gas flow, it is difficult for the converter department alone to provide the acid plant with a feed gas of sufficiently stable and strong SO_2 concentration (minimum 3.5-4.0 percent volume). The operating cycles of two or more converters must be synchronized to give the overall converting process a relatively continuous gas output. Sophisticated gas removal, control, and monitoring systems are required to achieve these results.

- Improve cleaning and treating of particulates in converter gas streams to allow better operation of SO_2 removal systems such as contact sulfuric acid plants. Converter gases must be cleaned of particulate matter prior to SO_2 control; this is accomplished in both dry and wet control devices, the latter of which are an integral part of the acid plant. Effective particulate removal is essential because settled dusts can block the active catalyst surface, decreasing the SO_2-SO_3 conversion rate. The dusts fill the spaces between the catalyst grains, increasing the rate of gas flow and reducing contact time. The quantity of gases processed drops because of the increase in flow resistance. Dusts that remain in the acid circuit contaminate the product and can form slimes that will clog nozzles and impair heat exchange. Monitoring and control devices can be contaminated, and there can be increased wear of pumps, blowers, and pipelines, as well as accelerated corrosion caused by heat imbalances. Production stops will then be necessary to regenerate the catalyst or repair or replace worn or damaged equipment. The maximum dust concentration in the gases entering the contact apparatus is 0.02 g/Nm^3. In addition, no arsenic or other catalyst poisons should be present. If these requirements are met, the contact apparatus should have an effective life of about 2 years, which is equivalent to a typical acid plant using a pyrite or elemental sulfur feed.

- Show how the procedures and results that are developed could be applied to the various types of copper smelters encountered in industry. At copper smelters where only converter gases are processed in the acid plant, there must be at least three converters, two of which are under blow while the third is undergoing maintenance, serving as a holding furnace, or on standby. The working converters must operate according to a precisely synchronized schedule to ensure the proper gas conditions. The scheduling of these operations is facilitated with a greater number of converters. Mathematical models of the converting process have been developed, and automatic controls for operation of the converter and gas removal system have been installed at a number of smelters in addition to those already installed on the Polish smelters. These include the Sociedad Mineria El Teniente smelter in Caletones, Chile, and the Norddeutsche Affinerie plant in Hamburg, Germany. Automatic control systems have thus been installed on both Peirce-Smith and Hoboken converters of various sizes. The Polish experience with automated process control should provide data potentially applicable to any modern copper smelter. However, an automatic gas removal system based upon blast intensity is expected to have more satisfactory results with siphon converters because of the reduced air infiltration.

SECTION 2

CONVERTER OPERATING PROCEDURES

BACKGROUND

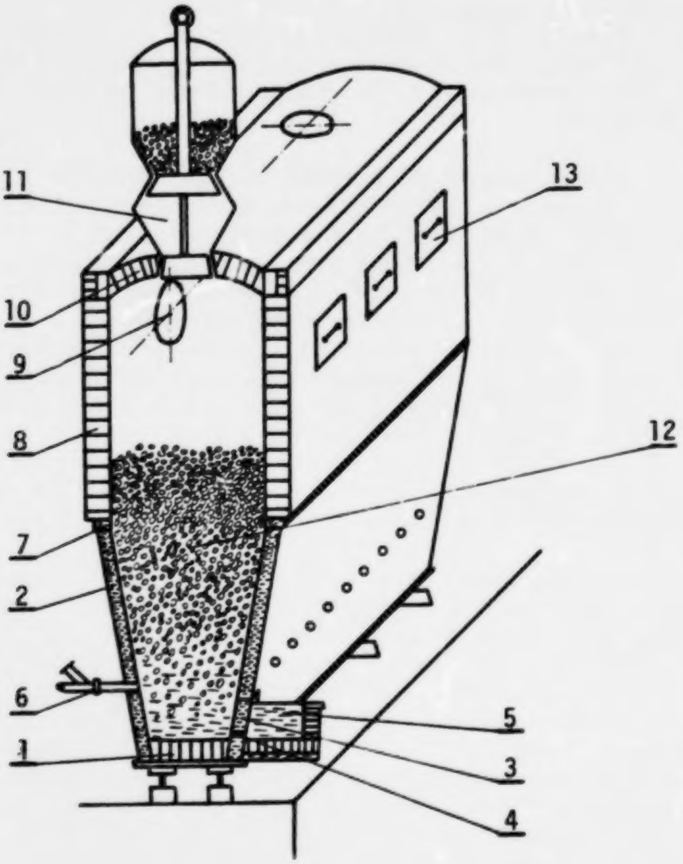
The mining and processing of copper ores and concentrates in Poland did not begin until after World War II. Because the original deposits mined were small and of low grade, the industry's growth was at first severely limited. The discovery of new ore deposits in 1957 permitted the development of a modern copper industry, whose production now more than meets Poland's domestic needs, allowing a surplus to be exported. The mines, concentrators, smelters, and associated facilities are all located in the mining districts in western Poland.

Mined ores are first upgraded by flotation to produce a concentrate which is low in iron and sulfur content and which contains from 20 to 25 percent copper. Table 1 presents an example of a typical concentrate composition. After arrival at the smelter, the concentrates are mixed with sulfite liquor, then dried in a rotary dryer and briquetted. The briquettes, which contain 19 to 24 percent copper, are charged to a rectangular water-jacketed shaft smelting furnace through double bell valves. Figure 1 is a diagram of the shaft furnace. The briquettes comprise about 80 percent of the furnace charge, with the remainder about equally divided between coke and recycled converter slag. Shaft furnace smelting is a continuous reduction process with heat provided by partial combustion from air blown through nozzles near the bottom of the furnace. Liquid products drain continuously from the furnace and into a settling tank. The liquid matte and slag separate in the tank because of their different specific gravities and mutual insolubility. Matte is tapped from the tank into 21-Mg ladles and transported to the converter aisle by overhead cranes. Exhaust gases from smelting exit the upper part of the furnace and pass through particulate control devices.

The copper matte obtained from smelting contains 55 to 67 percent copper, 6 to 14 percent iron, 19 to 23 percent sulfur, 1.5 to 3.0 percent lead, and 0.3 to 0.8 percent zinc. Matte composition may fluctuate because of variations in the concentrates and in the operating conditions of the shaft furnaces.

TABLE 1. TYPICAL CONCENTRATE COMPOSITON

Species	Concentration %
Cu	25.24
Fe	2.00
Pb	1.40
S	8.22
SO_4	0.08
CaO	8.64
MgO	5.10
Al_2O_3	6.28
SiO_2	17.80
Ag	0.041
CO_2	12.40
Organics	8.23
Other	4.5



1 - FURNACE BOTTOM
2 - SIDE AND FRONT WATER JACKETS
3 - WATER JACKET, CENTRAL TAP
4 - CENTRAL TAP
5 - SIPHON TAPPING SPOUT
6 - AIR NOZZLES
7 - WATER-COOLED SUPPORT CONSTRUCTION
8 - SHAFT
9 - GAS OFFTAKE
10 - FURNACE ROOF
11 - DOUBLE BELL VALVES
12 - CHARGE MIXTURE
13 - MANHOLES

Figure 1. Shaft furnace.

Siphon converters were chosen for use in the Polish copper industry for technical, economic, and environmental reasons. When the first smelter was constructed, it was believed that the current technology was not sufficiently advanced to allow the operation of Peirce-Smith converters in a manner that would guarantee a steady supply of high-strength SO_2 gases to feed an acid plant. The Hoboken design appeared to ensure more efficient utilization of SO_2, while simultaneously minimizing atmospheric emissions. The design was purchased from Mechim, S.A., of Brussels, Belgium.

Figures 2 and 3 are diagrams of the Polish siphon converter. The design is essentially the same as a Peirce-Smith, with a mouth in the middle of the furnace for charging the feed materials and skimming or pouring the liquid products. A side flue, or siphon, is attached to one of the converter end walls. It is connected to the exhaust tower with cylindrical ductwork and a seal ring, both of which rotate with the converter. For weight stability, the siphon is balanced with a counterweight. The converter assembly is supported on concrete pillars and is equipped with a rotating mechanism similar to that of a Peirce-Smith. At Glogow Nos. 1 and 2, the inner volume of the furnace permits an actual blister production of about 80 Mg per operating cycle, depending on the copper content of the matte.

The siphon converters installed at these newer smelters incorporate a number of improvements over the original Hoboken design. They are more sturdily constructed, and quartzite flux is fed continuously to them through automatic weighers. Fully automatic tuyere punchers of Polish design, which are controlled by blast air pressure, are also installed (See Engineering and Mining Journal, February, 1979). A gate is provided below the siphon to allow removal of buildups without stopping the converting process or breaking the link to the gas removal system. The seal ring connecting the ductwork to the exhaust tower is also of Polish design, and an improved counterweight is used that reduces the strain on the bearings and motors and provides more stable operation.

Four siphon converters are installed at the Glogow No. 1 smelter. Except where indicated otherwise, all data in this report refer to this smelter. The scope of the process operations is detailed in Figure 4, and the accompanying Table 2 presents the compositional range of the more significant process streams.

THE CONVERTING PROCESS

Converting is a batch operation that consists of blowing the liquid copper matte from the shaft furnace with compressed air to

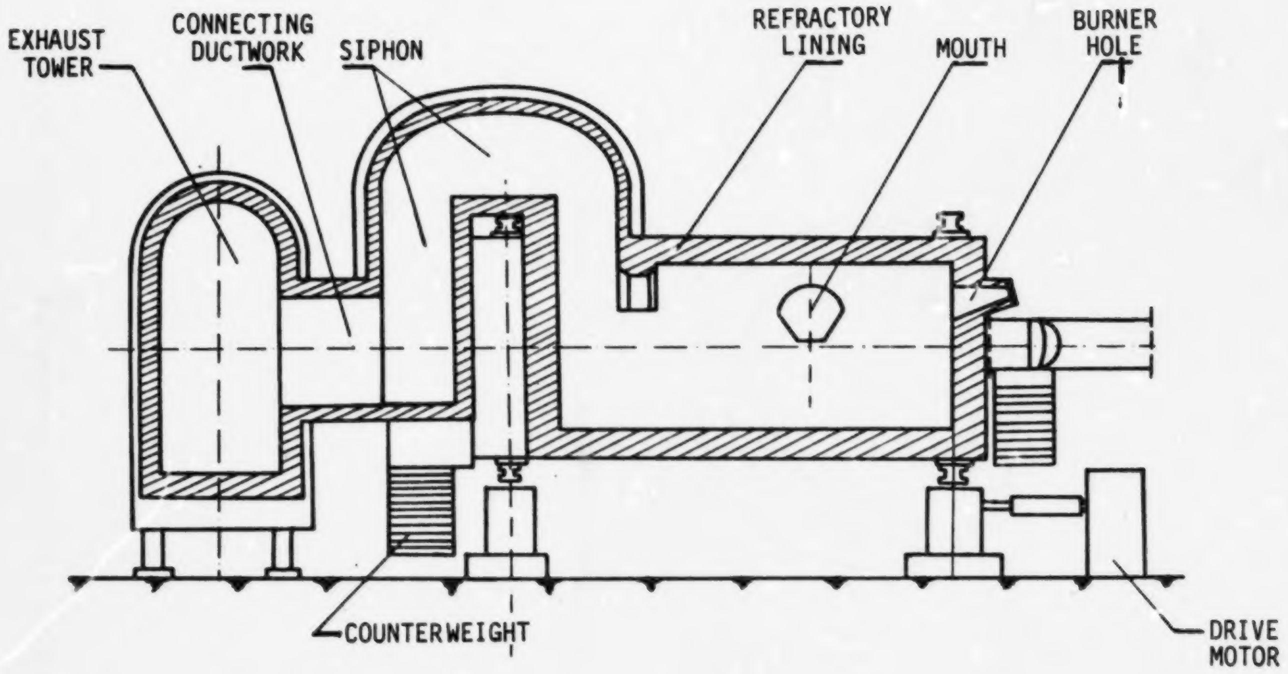

EXHAUST TOWER
CONNECTING DUCTWORK
SIPHON
REFRACTORY LINING
MOUTH
BURNER HOLE
COUNTERWEIGHT
DRIVE MOTOR

Figure 2. Siphon converter - longitudinal cross section.

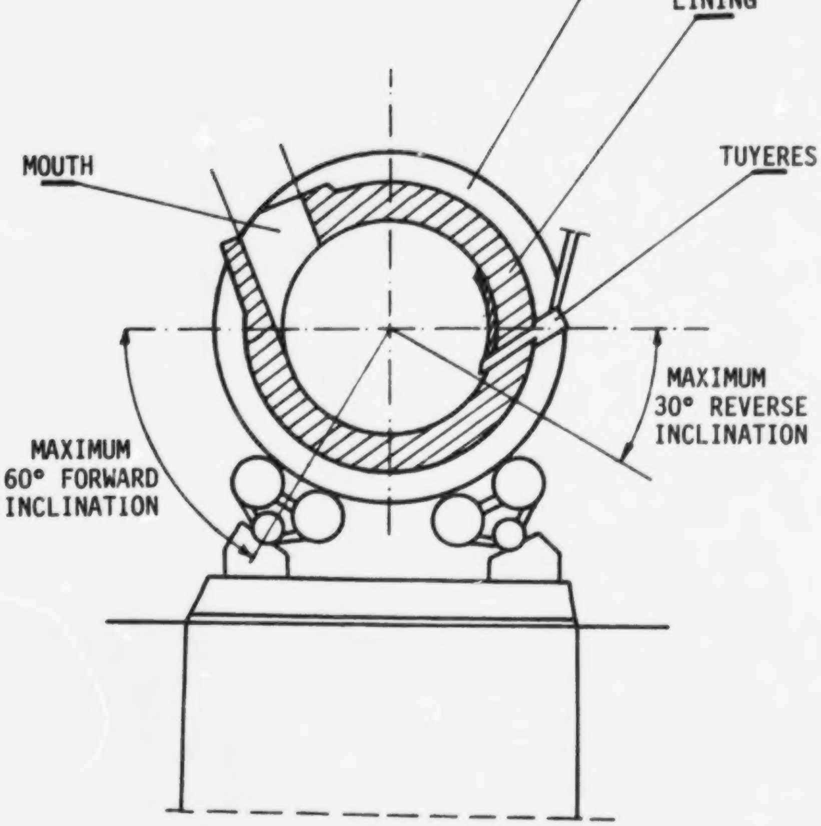
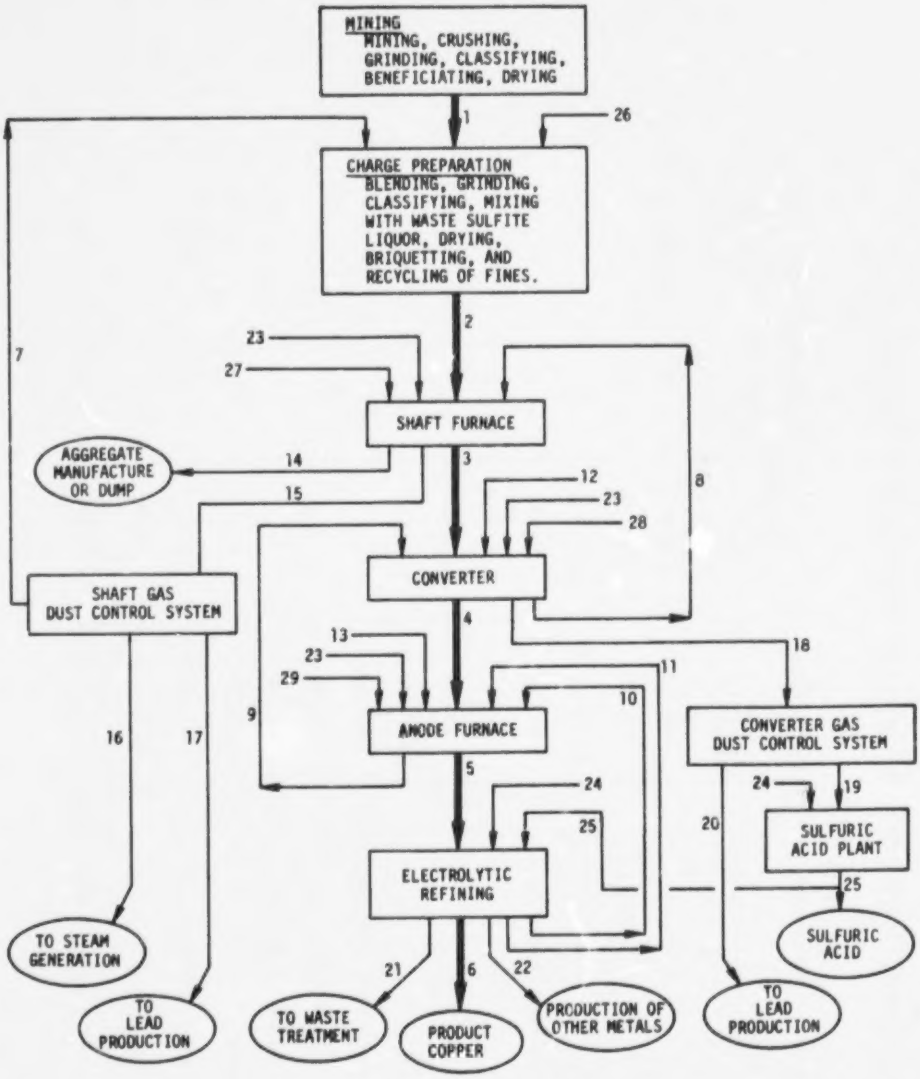

ROLLING RING
LINING
MOUTH
TUYERES
MAXIMUM 30° REVERSE INCLINATION
MAXIMUM 60° FORWARD INCLINATION

Figure 3. Siphon converter - cross section.


MINING
MINING, CRUSHING, GRINDING, CLASSIFYING, BENEFICIATING, DRYING
CHARGE PREPARATION
BLENDING, GRINDING, CLASSIFYING, MIXING WITH WASTE SULFITE LIQUOR, DRYING, BRIQUETTING, AND RECYCLING OF FINES.
SHAFT FURNACE
AGGREGATE MANUFACTURE OR DUMP
CONVERTER
SHAFT GAS DUST CONTROL SYSTEM
ANODE FURNACE
CONVERTER GAS DUST CONTROL SYSTEM
SULFURIC ACID PLANT
ELECTROLYTIC REFINING
SULFURIC ACID
TO STEAM GENERATION
TO LEAD PRODUCTION
TO WASTE TREATMENT
PRODUCT COPPER
PRODUCTION OF OTHER METALS
TO LEAD PRODUCTION
1
26
2
23
27
7
8
14
15
3
12
23
28
4
13
23
29
11
10
18
9
16
17
5
24
25
24
19
20
25
21
6
22

CUPRIFEROUS MATERIALS:

1. CONCENTRATE
2. BRIQUETTES
3. MATTE
4. BLISTER COPPER
5. ANODE COPPER
6. ELECGROLYTIC COPPER
7. SHAFT FURNACE DUSTS (COARSE)
8. CONVERTER SLAG
9. ANODE FURNACE SLAG
10. ANODE STUBS
11. RECLAIMED ELECTROLYTE COPPER
12. COPPER SCRAP TO CONVERTER
13. COPPER SCRAP TO ANODE FURNACE

BY-PRODUCTS AND WASTES:

14. SHAFT FURNACE SLAG
15. SHAFT FURNACE OFF-GASES
16. CLEAN SHAFT FURNACE GASES
17. SHAFT FURNACE DUSTS (FINE)
18. CONVERTER OFF-GASES
19. CLEAN CONVERTER GASES
20. CONVERTER DUSTS
21. WASTE ELECTROLYTE
22. ANODE SLIME

NON-CUPRIFEROUS INPUTS:

23. BLAST AIR
24. WATER
25. SULFURIC ACID
26. WASTE SULFITE LIQUOR
27. COKE
28. QUARTZITE
29. WOOD AND OIL

Figure 4. Flowsheet - Glogow No. 1 smelter.

TABLE 2. INPUTS AND PRODUCTS OF COPPER SMELTING - GLOGOW NO. 1

Number referenced to Fig. 4	Cu	Analysis, %		Organics
		Pb	S	
1 Concentrate	20-25	0.7-2.0	6-11	3.5-7.0
2 Briquettes	19-24	1.0-3.0	7-12	4.0-6.5
3 Matte	55-67	1.5-3.0	19-23	-
4 Blister	98.5-99.0	0.1-0.3	-	-
7 Shaft furnace dusts (coarse)	13-21	5-15	6-12	8-18
8 Converter slag	5-8	0.5-1.0	-	-
9 Anode furnace slag	40-60	0.5-1.0	-	-
10 Anode discards	99.2-99.3	0.1-0.2	-	-
11 Copper from electrolyte purification	53-78	-	-	-
12 Copper scrap to converter	60-99	-	-	-
13 Copper scrap to anode furnace	98-99	-	-	-
14 Shaft furnace slag	0.3-0.5	0.2	-	-
17 Shaft furnace dusts (fines)	0.4-3.0	35-45	7-14	6-13
20 Converter dusts	1.0-2.5	45-55	6-14	-
22 Anode slime	1.5-2.0	20-45	-	-

oxidize the sulfides and convert the remaining iron into a slag. Three distinct stages of converter operation are used in the Polish smelters:

1. Iron sulfide oxidation with siliceous fluxing materials, forming an iron oxide slag and sulfur oxides (slag blow)

   $$2FeS + 3O_2 + SiO_2 \rightarrow (FeO)_2 \cdot SiO_2 + 2SO_2$$

   $$3FeS + 5O_2 \rightarrow Fe_3O_4 + 3SO_2$$

2. Copper sulfide oxidation, forming copper and sulfur dioxide (copper blow)

   $$Cu_2S + O_2 \rightarrow 2Cu + SO_2$$

3. Partial copper oxidation to further lower the lead content (finish blow)

   $$2Cu + 1/2O_2 \rightarrow Cu_2O$$

   $$Cu_2O + Pb \rightarrow PbO + 2Cu$$

Additional reactions occur in the first two stages, including the oxidation of lead and zinc sulfides to oxides and sulfates and partial oxidation of SO_2 to SO_3.

The product of converting is blister copper, which has a typical analysis of 98.5 to 99.0 percent copper, 0.1 to 0.3 percent lead, and 0.4 to 0.6 percent oxygen. The blister copper is transported in ladles either directly to a fire-refining furnace or to another converter in which it may be held in a molten state. The converter slag contains up to 50 percent iron, 24 to 26 percent silica and 5 to 8 percent copper, and its quantity is 21 to 23 percent of the input matte. The slag is recycled to the shaft furnace to recover its copper content. The high iron content of the slag also improves the quality of shaft furnace slag, about 60 percent of which is used in the cement industry and as trap rock in roadbeds. The high concentration of copper in the slag occurs because the blast is not shut off during slag pouring, but only reduced in intensity; this is done to ensure a more continuous feed of gas to the acid plant. The converter off-gases, containing a high SO_2 and SO_3 content, are withdrawn from each converter by individual fans to provide the control necessary to minimize both excessive dilution and fugitive emissions. The gases are cooled and cleaned of particulate before they are used for acid production. Dusts from the dry particulate control equipment (ESP's) are rich in lead and zinc sulfates; a typical analysis is 45 to 55 percent lead, 4 to 8 percent zinc, and 6 to 14 percent sulfur. They are further processed to recover the lead values.

In addition to liquid matte, a converter is charged with silica flux and cold recycled copper-bearing materials from within the smelter. The flux is quartzite or quartzite gravel with a grain size of 10 to 30 mm, containing 92 to 95 percent SiO_2, 1.5 to 5.0 percent Al_2O_3, and 1 to 2 percent iron. The amount of flux to be charged to each operating cycle is calculated based on the matte analysis; the intent is to convert all of the iron in the matte to the silicate $(FeO)_2SiO_2$. For example, when the iron content in the matte is 9.72 percent and the average silica content in the flux is 93 percent, 56.2 kg of dry flux is needed per Mg of matte. The use of cold recycled materials prevents excessive temperature rises in the converter and thus prolongs lining life; it also permits the recovery of this additional copper. The recycled materials include slag from the refining furnaces (which has an analysis of 40 to 60 percent copper, 5 to 10 percent iron, 15 to 25 percent SiO_2, and 0.5 to 1.5 percent lead), and waste materials from the converter department such as skulls, slag, and accretions.

In spite of the high copper content of the matte charged to the converter, the process is autogenous and the heat released within the furnace is sufficient to compensate for heat losses to the atmosphere as well as that necessary to melt the cold recycled materials. The exothermic reaction of the iron and oxidation of copper create the heat, and a relatively high blast intensity with maximum oxygen utilization is possible.

Blast air is supplied to the converter up to 30,000 Nm^3/h at a pressure of 8300 to 12,400 kg/m^2; variations result from the number of tuyeres used and their degree of obstruction. From 38 to 40 tuyeres are normally used, with 4 to 6 kept in reserve. The tuyeres are cleaned by automatic punchers, which may also be overridden manually.

The theoretical air requirements and time for converting one Mg of matte, assuming total oxygen utilization and medium blast intensity (23,000 Nm^3/h), are shown below:

Slag blow	258.50 Nm^3	0.674 min
Copper blow	541.00 Nm^3	1.411 min
Finish blow	12.67 Nm^3	0.033 min
	812.17 Nm^3	2.118 min

In practice, however, both air requirements and operating time are greater than the theoretical because of air losses at the tuyeres and incomplete oxygen use within the converter.

In the siphon converter, the mouth is open at all times, whereas in the Peirce-Smith, a gas collection hood covers the converter mouth. Because the mouth is open, the efficient operation and control of the siphon off-take is essential. Improper operation can result in either a pressure increase in the converter that allows particulate and gases to escape through the open mouth, or an excessive inflow of air which causes a carryover of substantial amounts of materials (slag, matte, and blister) into the siphon system. Solids drawn into the siphon form deposits that impede the gas flow. Two operating factors are of particular importance in preventing the carryover of these materials. First, the converter should only be charged when the surface of the melt is 0.2 to 0.3 m below the horizontal axis. Second, only moderate blast intensities (about 12 Nm^3/min) should be applied at each tuyere. The other operating requirements for siphon converters are similar to those for the Peirce-Smith design. These include holding operating temperatures below 1350°C, use of high quality refractory materials, a careful program of maintenance (especially tuyere cleaning), proper addition of dry flux, and maximum utilization of the operating time under blast.

Assuming total oxygen utilization, the theoretical volume and SO_2 content of converter gases are as follows:

Slag blow - 240.2 Nm^3/Mg matte, 14.99 percent SO_2

Copper blow - 539.3 Nm^3/Mg matte, 20.83 percent SO_2

In actual practice, the gas volume is about twice and the SO_2 content about half of the theoretical because of incomplete oxygen use and air inleakages into the furnace and ductwork.

THE OPERATING CYCLE

A typical complete operating cycle for a single converter includes the processing of 147 Mg of liquid matte (seven 21-Mg ladles), 9 Mg of solid recycled materials, and 8.5 Mg of flux. The operating cycle is composed of the following distinct operations:

- The solid recycled materials are transferred from the stockyard and converter aisle in ladles and scoops and charged to the converter by the overhead crane.
- Liquid matte from the shaft furnace is charged to the converter by the overhead crane in 21-Mg ladles. Before pouring the first ladle, the converter is tilted 10 degrees toward the tapping position, and a blast of

5000 Nm^3/h is put on to prevent the tuyeres from clogging. After pouring the matte, the blast is shut off. This practice is continued through the pouring of the first four ladles, which are supplied to the converter every 5 to 10 minutes. During this period, the off-gases, which contain 1 to 2 percent SO_2, are released with natural draft through a bypass into an emergency stack.

- When the fifth ladle of matte has been poured into the converter, the full blast is put on, the emergency stack bypass is closed, and the converter is tilted back to its vertical operating position. This begins the slag blow. After 10 minutes under blast, a sixth ladle of matte is poured into the converter and flux is added in three batches (1.8 Mg each). This first stage of the converting process operates with a blast intensity of approximately 20,000 to 27,500 Nm^3/h, with a pressure at the tuyeres of about 8300 to 12,400 kg/m^2. During this stage, the blast rate and pressure are unstable because of tuyere blockage, and regular punching is necessary to reduce this problem. The converting process is run continuously until nearly all iron sulfide in the matte has oxidized and liquid slag is produced. This process is monitored by observation of the matte and slag layer on the tuyere cleaning punches.

- When the matte has oxidized to a sufficient degree, the converter is rolled out and the slag is drained into a ladle. During this pouring operation, the blast is not shut off entirely, but is instead reduced to about 10,000 Nm^3/h.

- After the slag pour, the converter is tilted back to its vertical operating position, and the blast intensity is simultaneously returned to its previous level. A seventh ladle of matte is added, followed by 1.1 Mg of flux, and the slag blow is continued.

- When the matte has again oxidized sufficiently, the slag is poured off. As before, the blast is reduced in intensity during pouring.

- After the second slag pour, the second stage of the converting process, the copper blow, begins. This process is run continuously at a blast intensity of approximately 22,000 to 26,000 Nm^3/h and a tuyere pressure of about 8300 to 12,400 kg/m^3 until nearly all of the copper sulfide has been oxidized. The blast parameters in this stage are more stable than during the slag blow. The end of the copper blow is determined by observation of the blister on the cleaning punches.

- The final converting stage is the finish blow, during which the blister is partially oxidized to decrease its lead content. This stage is run at a decreased blast intensity (approximately 20,000 Nm^3/h) for 5 to 10 minutes.

- After completion of the finish blow, the converter is rolled forward, the blast and gas off-take to the particulate control system are shut off, and the emergency stack bypass is reopened. The bath stands for a short time to mature, and a final slag is drained from the copper surface. A 2-Mg batch of flux is then added to fix the remaining slag, which is difficult to remove from the copper. This flux remains in the converter aisle and is reused in the next operating cycle.

- The ladle is then changed and the blister is poured off in batches of approximately 25 Mg each. The ladles are transferred to the refining furnace by the overhead crane. If the refining furnace is not ready to accept the blister, it is either held in the converter or transferred into another converter, which acts as a holding furnace, operating as in the copper blow, with supplemental fuel if necessary.

- The converter is prepared for its next operating cycle by such operations as tuyere maintenance and breaking off buildups around the mouth.

A THREE-CONVERTER SYSTEM

To assure steady utilization of matte from the shaft furnace and a continuous SO_2 supply to the acid plant, individual converter operating cycles are synchronized to overlap the periods of operation under blast. An operating schedule of a three-converter system (K1, K2, and K3) is presented here based on measurements made over five successive operating cycles of the K1 converter. The amount of matte processed in each converter and operating cycle was 147 Mg, and its composition was stable and within the typical analysis given previously in this section. Measurements were made of the duration of each operating stage, the blast intensity and gas flow to the particulate control system, and the SO_2 and SO_3 concentrations in the combined gases from all three converters. The duration under blast was measured separately for each operating stage. Table 3 shows the proportion of the total operating cycle during which each converter was

TABLE 3. CONVERTER TIME UNDER BLAST

Cycle	Operations					
	Charging and waiting for matte (min)	Time under blast (min)			Matte pouring, preparation for next cycle (min)	Blast time/ total cycle (%)
		Slag blow	Copper blow	Finish blow		
Converter K1:						
1	45	117	235	7	120	68.5
2	60	114	230	8	125	65.5
3	50	120	242	7	110	69.7
4	55	115	238	7	125	66.7
5	50	118	233	7	117	68.2
Converter K2:						
1	42	112	228	8	120	68.2
2	65	115	230	7	108	67.0
3	40	110	236	7	132	67.2
4	55	118	225	8	119	66.9
5	45	110	240	7	123	68.0
6	35	116	227	7	137	67.0
Converter K3:						
1	52	116	232	6	104	69.4
2	40	118	236	7	124	68.8
3	45	122	230	7	116	69.0
4	35	120	226	7	110	70.9
5	47	113	244	8	120	68.6
6	55	115	235	8	122	66.9

the slag blow. The end of the copper blow is determined by observation of the blister on the cleaning punches.

- The final converting stage is the finish blow, during which the blister is partially oxidized to decrease its lead content. This stage is run at a decreased blast intensity (approximately 20,000 Nm^3/h) for 5 to 10 minutes.
- After completion of the finish blow, the converter is rolled forward, the blast and gas off-take to the particulate control system are shut off, and the emergency stack bypass is reopened. The bath stands for a short time to mature, and a final slag is drained from the copper surface. A 2-Mg batch of flux is then added to fix the remaining slag, which is difficult to remove from the copper. This flux remains in the converter aisle and is reused in the next operating cycle.
- The ladle is then changed and the blister is poured off in batches of approximately 25 Mg each. The ladles are transferred to the refining furnace by the overhead crane. If the refining furnace is not ready to accept the blister, it is either held in the converter or transferred into another converter, which acts as a holding furnace, operating as in the copper blow, with supplemental fuel if necessary.
- The converter is prepared for its next operating cycle by such operations as tuyere maintenance and breaking off buildups around the mouth.

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Table 4 shows the characteristics of the combined gases fed to the acid plant from the three converters during the test period. Based upon these data, the number of converters under blast during the test period was as follows:

3 converters	212 min	7.98 percent
2 converters	2391 min	90.06 percent
1 converter	52 min	1.96 percent
0 converters	0 min	0
	2655 min	100.00 percent

It is evident from Table 4 that a system of three converters can be optimized to assure a continuous gas supply to the acid plant, with the simultaneous operation of two converters under blast occurring 90.06 percent of the time. The total combined gas stream ranged from 1400 to 1600 Nm^3/min for 96.4 percent of the test period, and from 900 to 1000 Nm^3/min for the remainder. The SO_2-SO_3 concentration remained within the 7.0 to 8.8 percent range for 94.4 percent of the test and was higher (up to 11.5 percent) for the remainder of the time.

Table 5, extrapolated from Table 4, shows that there were six combinations of converter operating cycles during the test. During nearly 90 percent of the test period, there were either two converters in the copper blow or one each in the slag and copper blows.

TABLE 4. GAS CHARACTERISTICS IN A THREE-CONVERTER SYSTEM

	Time (min)	Converter K1[a]		Converter K2[a]		Converter K3[a]		Combined gas (Nm^3/min)	Avg. SO_2 and SO_3 conc. (%)
		Cycle	Blow	Cycle	Blow	Cycle	Blow		
1	18	-	-	1	C	1	S	1400	8.2
2	27	-	-	1	C	1	C	1600	8.8
3	7	1	S	1	C	1	C	1600	11.6
4	8	1	S	1	F	1	C	1400	8.1
5	102	1	S	-	-	1	C	1400	8.1
6	83	1	C	-	-	1	C	1600	8.7
7	5	1	C	2	S	1	C	1600	11.5
8	6	1	C	2	S	1	F	1400	8.1
9	104	1	C	2	S	-	-	1400	8.1
10	37	1	C	2	C	-	-	1600	8.7
11	3	1	F	2	C	-	-	1000	7.0
12	4	1	F	2	C	2	S	1400	8.1
13	114	-	-	2	C	2	S	1400	8.1
14	66	-	-	2	C	2	C	1600	8.7
15	6	2	S	2	C	2	C	1600	11.7
16	7	2	S	2	F	2	C	1400	8.4
17	101	2	S	-	-	2	C	1400	8.3
18	47	2	C	-	-	2	C	1600	8.9
19	9	2	C	3	S	2	C	1600	11.8
20	7	2	C	3	S	2	F	1400	8.4
21	94	2	C	3	S	-	-	1400	8.4
22	73	2	C	3	C	-	-	1600	8.7
23	2	2	F	3	C	-	-	1000	6.9
24	6	2	F	3	C	3	S	1400	7.9
25	116	-	-	3	C	3	S	1400	7.9
26	39	-	-	3	C	3	C	1600	8.7
27	7	-	-	3	F	3	C	1000	7.0
28	13	-	-	-	-	3	C	900	7.8
29	120	3	S	-	-	3	C	1400	8.1
30	51	3	C	-	-	3	C	1600	8.6
31	3	3	C	-	-	3	F	1000	6.7
32	4	3	C	4	S	3	F	1400	7.9
33	114	3	C	4	S	-	-	1400	7.9
34	37	3	C	4	C	-	-	1600	8.7
35	33	3	C	4	C	4	S	1600	11.4
36	7	3	F	4	C	4	S	1400	8.2
37	80	-	-	4	C	4	S	1400	8.2
38	68	-	-	4	C	4	C	1600	9.0
39	8	-	-	4	F	4	C	1000	7.2
40	9	-	-	-	-	4	C	900	8.0
41	115	4	S	-	-	4	C	1400	8.3
42	26	4	C	-	-	4	C	1600	8.7
43	7	4	C	-	-	4	F	1000	6.8
44	7	4	C	-	-	-	-	900	7.6
45	110	4	C	5	S	-	-	1400	8.2
46	40	4	C	5	C	-	-	1600	8.5
47	48	4	C	5	C	5	S	1600	11.3
48	7	4	F	5	C	5	S	1400	8.1
49	58	-	-	5	C	5	S	1400	8.1
50	87	-	-	5	C	5	C	1600	8.4
51	7	-	-	5	F	5	C	1000	6.6
52	23	-	-	-	-	5	C	900	7.4
53	118	5	S	-	-	5	C	1400	7.9
54	9	5	C	-	-	5	C	1600	8.5
55	8	5	C	-	-	5	F	1000	7.0
56	116	5	C	6	S	-	-	1400	8.1
57	59	5	C	6	C	-	-	1600	8.8
58	41	5	C	6	C	6	S	1600	11.6
59	7	5	F	6	C	6	S	1400	8.3
60	67	-	-	6	C	6	S	1400	8.3
61	50	-	-	6	C	6	C	1600	8.8

[a] S = slag blow, C = copper blow, F = finish blow.

TABLE 5. OPERATION OF THREE CONVERTERS UNDER BLAST

Converters under blast	Blow*	Duration	
		Minutes	Percent
2	S, C	1547	58.3
2	C, C	799	30.1
3	S, C, C	149	5.6
3	S, C, F	63	2.4
2	C, F	45	1.7
1	C	52	1.9

[*] S = slag blow, C = copper blow, F = finish blow.

The production of sulfuric acid by the contact process requires a stable gas stream of steady and strong SO_2 concentration. Some fluctuation in feed gas is unavoidable, however, because the individual converters are operating at different stages of blowing and pouring or charging. The converter exhaust fans must be regulated to minimize the differences in quantity and composition to produce a stable combined gas. Table 6 shows the fan drafts applied to the individual converters at different operating stages during the test period. It shows that a constant amount of draft was applied during slag or finish blows, but that the amount of draft during a copper blow was dependent upon the operating status of the other converters.

TABLE 6. FAN DRAFT ON INDIVIDUAL CONVERTERS IN A THREE-CONVERTER SYSTEM

Blow	Other converters under blast		Fan draft (Nm^3/min)
	Number	Blow*	
Slag		-	600
Copper	0	-	900
	1	S	800
	1	C	800
	1	F	700
	2	S, C	500
	2	S, F	500
Finish		-	300

[*] S = slag blow, C = copper blow, F = finish blow.

DILUTION OF CONVERTER OFF-GASES

The quantity of gases released by the converting process depends on the amount of blast air and on chemical reactions between oxygen and the charge in the converter. The off-gases, however, are diluted by air which is drawn in both through the mouth and through leaks in the gas pumping, cooling, and particulate removal system. It is difficult to measure directly the amount of air dilution, although indirect estimates can be made. Strip charts from continuous recording flow meters located on the tuyere headers were used to determine the average blast intensities. Based on these, and on process duration, the theoretical volume of gases generated in each operating cycle was derived by planimetry. Loss of 3 percent air at the tuyeres and utilization of 85 to 92 percent of the oxygen were assumed, based on indications from other tests. Results of those calculations are presented in Table 7.

The effects of the oxygen/sulfur reactions on the quantity of off-gases generated as converting proceeds can be illustrated as follows:

- Slag blow:

$$
\begin{aligned}
2FeS + \boxed{3O_2} + SiO_2 &\rightarrow (FeO)_2SiO_2 + \boxed{2SO_2}\\
3FeS + \boxed{5O_2} &\rightarrow Fe_3SO_4 + \boxed{3SO_2}
\end{aligned}
$$

$$\frac{\text{off-gases}}{\text{blast air}} = 0.90$$

- Copper blow:

$$
\begin{aligned}
CuS + \boxed{O_2} &\rightarrow 2Cu + \boxed{SO_2}\\
2ZnS + \boxed{3O_2} &\rightarrow 2ZnO + \boxed{2SO_2}\\
3PbS + \boxed{3O_2} &\rightarrow 2PbO + \boxed{2SO_2}
\end{aligned}
$$

$$\frac{\text{off-gases}}{\text{blast air}} = 0.96$$

- Finish blow:

$$4\,Cu + \boxed{O_2} \rightarrow 2Cu_2O$$

$$\frac{\text{off-gases}}{\text{blast air}} = 0.76$$

TABLE 7. CONVERTER OFF-GAS PRIOR TO DILUTION

Cycle	Average blast intensity (Nm^3/min)			Gas volume (Nm^3/min)		
	Slag blow	Copper blow	Finish blow	Slag blow	Copper blow	Finish blow
Converter K1:						
1	376.9	392.8	300.0	339.2	377.1	234.0
2	385.1	396.5	268.7	346.6	380.6	209.6
3	376.7	387.8	285.7	339.0	372.3	222.8
4	388.7	390.8	314.3	349.8	375.2	245.2
5	377.5	398.3	307.1	339.7	382.4	239.5
Converter K2:						
1	412.9	426.3	275.0	371.6	409.2	214.5
2	399.1	410.9	321.4	359.2	394.5	250.7
3	407.7	394.7	314.3	366.9	378.9	245.2
4	382.2	424.2	287.5	344.0	407.2	224.2
5	418.2	401.2	314.3	376.4	385.2	245.2
6	383.2	409.2	321.4	344.9	392.8	250.7
Converter K3:						
1	360.3	377.6	341.7	324.3	362.5	266.5
2	358.9	387.7	300.0	323.0	372.2	234.0
3	355.7	392.4	300.0	320.1	376.7	234.0
4	366.2	401.8	292.9	329.6	385.7	228.5
5	380.5	375.0	262.5	342.4	360.0	204.7
6	371.3	380.0	268.7	334.2	364.8	209.6

Using these factors and analytical data of SO_2 concentration, the proportion of dilution air to total off-gas volume in each cycle was calculated, as shown in Tables 8 and 9. From estimates based on other analyses, it appears that about 30 to 40 percent of the dilution air comes through the mouth, and the remainder results from leaks.

OPERATING PROBLEMS

To obtain an off-gas composition approaching theoretical in a siphon converter, precisely interrelated operation of the blast intensity and gas removal system (fan draft) is required to yield a pressure difference that is essentially zero. This minimizes excessive air dilution through the mouth and other openings, while simultaneously allowing gas removal through the siphon. In practice, this goal is difficult to achieve because gas removal is controlled by changes in fan speed that are slower to respond than the rapid changes which can be made to blast intensity. The converter gases can be pulled away too rapidly, increasing dilution and thereby increasing gas volume and reducing SO_2 concentration. This also has the effect of greatly reducing fugitive gas emissions through the converter mouth. As the converter operates in this manner, however, liquids in the furnace are carried over into the siphon system and ductwork by the increased gas velocity. When gas velocity decreases inside this siphon and flow direction changes, these materials settle from the off-gases and form deposits in the exhaust system, further reducing its cross section and increasing the resistance to gas flow. In extreme cases, the siphon or ductwork become so clogged that despite increased fan speed, fugitive emissions occur through the converter mouth. This situation could be partially corrected by an appropriate reduction in blast intensity; however, this option is not often viable because of the thermal process parameters and the necessity for maintaining production. An increase in dilution also reduces the gas temperature and may result in the condensation of sulfuric acid in the ductwork.

The carryover of materials from the furnace into the siphon is the major drawback of the siphon converter design. Deposition is increased by the processing of a matte with high lead content, as this increases the dust loading in the off-gases. The accretions occur most rapidly when the converter is filled with matte above its horizontal axis and when it operates at high blast intensities (more than 13 Nm^3/min per tuyere). Removal of the deposits is difficult and can only be accomplished when the converter is cold to permit entry of men and equipment.

Disintegration of the refractory lining in the converter is most rapid near the tuyeres, even though, to compensate for this,

TABLE 8. CONVERTER GAS DILUTION DURING SLAG AND FINISH BLOWS

Cycle	Slag blow Fan draft 600 Nm^3/min		Finish blow Fan draft 300 Nm^3/min	
	Dilution air (Nm^3/min)	Dilution air/total off-gas (%)	Dilution air (Nm^3/min)	Dilution air/total off-gas (%)
Converter K1:				
1	260.8	43.5	66.0	22.0
2	253.4	42.2	90.4	30.1
3	261.0	43.5	77.2	25.7
4	250.2	41.7	54.8	18.3
5	260.3	43.4	60.5	20.2
Converter K2:				
1	228.4	38.1	85.5	28.5
2	240.8	40.1	49.3	16.4
3	233.1	38.8	54.8	18.3
4	256.0	42.7	75.8	25.3
5	223.6	37.3	54.8	18.3
6	255.1	42.5	49.3	16.4
Converter K3:				
1	275.7	45.9	33.5	11.2
2	277.0	46.2	66.0	22.0
3	279.9	46.6	66.0	22.0
4	270.4	45.1	71.5	23.8
5	257.6	42.9	95.3	31.8
6	265.8	44.3	90.4	30.1

TABLE 9. CONVERTER GAS DILUTION AT VARIOUS FAN DRAFTS DURING COPPER BLOWS

Fan draft Nm^3/min	Dilution air (Nm^3/min)				Dilution air/total off-gas (percent)			
	500	700	800	900	500	700	800	900
Converter K1:								
Cycle 1	122.9	-	422.9	-	24.6	-	52.9	-
Cycle 2	119.4	-	419.4	-	23.9	-	52.4	-
Cycle 3	127.7	327.7	427.7	-	25.5	46.8	53.5	-
Cycle 4	124.8	324.8	424.8	524.8	25.0	46.4	53.1	58.3
Cycle 5	117.6	317.6	417.6	-	23.5	45.4	52.2	-
Converter K2:								
Cycle 1	90.8	-	390.8	-	18.2	-	48.8	-
Cycle 2	105.5	305.5	405.5	-	21.1	43.6	50.7	-
Cycle 3	121.1	321.1	421.1	-	24.2	45.9	52.6	-
Cycle 4	92.8	-	392.8	-	18.6	-	49.1	-
Cycle 5	114.8	-	414.8	-	23.0	-	51.8	-
Cycle 6	107.2	-	407.2	-	21.4	-	50.9	-
Converter K3:								
Cycle 1	137.5	-	437.5	-	27.5	-	54.7	-
Cycle 2	127.8	-	427.8	-	25.6	-	53.5	-
Cycle 3	-	323.3	423.3	523.3	-	46.2	52.9	58.1
Cycle 4	-	314.3	414.3	514.3	-	44.9	51.8	57.1
Cycle 5	-	340.0	440.0	540.0	-	48.6	55.0	60.0
Cycle 6	-	-	435.2	-	-	-	54.4	-

the lining is thickest in this area. Refractory wear is caused by the effects of high temperatures, abrasion from the charge materials, and chemical reactions.

OPERATING COST EVALUATION

The Glogow No. 1 smelter operates with three daily 8-hour shifts. Three workers, a first furnaceman (converter operator) and two assistants, are directly responsible for each converter. Two additional workers are indirectly associated with each converter in such supporting functions as materials transfer. Other associated staff include electricians and control room personnel (1.2 man per converter), laboratory technicians who perform material analyses (0.3 man per converter), and the operators of the particulate control system (1.2 man per converter). One worker is required for general maintenance and cleanup at each converter on the first shift, and two are needed each shift for maintenance of the particulate control equipment (with a third assigned to each operating converter on the first shift). Supervisory and management personnel are not included in these requirements.

Electrical usage is recorded throughout the smelter but not for each department, so only estimated requirements can be given for each process. The total estimated electrical consumption for the converter aisle and its particulate control system in 1976 and 1977 was 104 kWh/Mg blister product. The approximate distribution according to individual need was as follows:

Blast air compression	50%
Converter rotation	3%
Gas cooling	3%
Gas pumping	36%
Particulate control system	8%

Other utility requirements include natural gas, water, and compressed air. Natural gas firing (heating value - 6500 kcal/ Nm^3) is used both to reheat a cold converter and to allow a converter to act as a holding furnace. Gas consumption is approximately 486 Nm^3 per Mg blister copper. Water is used to purge the bearings of the exhaust fans, as well as other rotating equipment, and for general maintenance; water requirements are 1.33 m^3 per Mg blister copper. Compressed air (six atmospheres) is used for driving the tuyere punchers, operating jackhammers and other equipment used for maintenance and refractory replacement, cleaning the particulate control system, and controlling servo-motors and monitoring equipment. The consumption of compressed air is 20.8 m^3 per Mg blister copper.

SECTION 3

THE GAS CLEANING SYSTEM

CONVERTER EMISSION CHARACTERISTICS

It is difficult to fully analyze the physical and chemical properties of the gases leaving the siphon converter because of their high temperatures, particulate loading, and chemical reactivity. Some parameters have been directly measured, while others, such as gas volume, have been determined indirectly as described in Section 2. The test methods and equipment used are described in Appendix C of this report.

The total volume of the off-gas after it leaves the gas cooler on each converter ranges from 22,000 to 50,000 Nm^3/h at typical blast intensities; the amount of dilution is affected by the condition of the siphon system, coolers, and connecting ductwork. The operating temperature in the siphon converters does not rise above 1350°C. The temperature of the off-gas before it enters the gas coolers generally ranges between 800° and 1050°C; it is primarily dependent on the blast intensity, blowing stage, and gas dilution. The gas temperature is approximately 50° to 100°C higher during the copper blow than the slag blow. When the blast is shut off or reduced, temperature drops to 600° to 650°C. The temperatures measured at other points in the gas offtake system are as follows:

gas cooler outlet	350° to 500°C
first-stage fan	450°C (maximum)
dirty gas mixing chamber	300° to 450°C
ESP inlet	280° to 400°C
ESP outlet	270° to 350°C
clean gas mixing chamber	300°C (average)
ductwork to acid plant	150° to 250°C

As detailed in Section 2, the theoretical SO_2 content of the converter gas (assuming total oxygen utilization) is 14.99 percent SO_2 for the slag blow and 20.83 percent SO_2 for the copper blow. Monitoring and analysis of the gas before it enters the coolers show approximately 13 percent SO_2 during the slag blow and 18 percent SO_2 during the copper blow. Taking into account dilution air, the concentration in the gases after leaving the cooler should equal 7.9 to 9.4 percent SO_2 (8.7 percent average) for the slag blow, and 8.3 to 17.1 percent SO_2 (12.7 percent average) for the copper blow, with an overall average of 11.4 percent SO_2.

Since the gas temperature does not exceed 1300°C, formation of SO_3 results primarily from catalytic oxidation of SO_2. Many substances are either true catalysts or in other ways promote SO_2 oxidation; some that have been found in the converter dusts are listed in Table 10.

TABLE 10. SO_2 CATALYSTS FOUND IN SIPHON CONVERTER DUSTS

Catalyst	Reaction temperature °C	SO_2 converted %
Fe_2O_3	625	69.5
$Fe_2O_3 + Bi_2O_3$	625	72.5
$Fe_2O_3 + SnO_2$	600	76.2
CuO	700	58.7
As_2O_3	670	55.0

Studies indicate that SO_3 formation results largely from the reaction of SO_2 with iron oxides present in the dusts entrained in the gas stream, in accordance with reactions such as the following:

$$2FeS + 3O_2 + SiO_2 \rightarrow (FeO)_2SiO_2 + 2SO_2$$

$$3FeS + 5O_2 \rightarrow Fe_3O_4 + 3SO_2$$

$$4Fe_3O_4 + O_2 \rightleftarrows 6Fe_2O_3$$

This occurs principally during the slag blow, when the quantity of iron in the converter dusts is greatest. The surface of contact between the dusts and gases is also much larger with very fine dusts. For example, the surface area of dusts 6 μm in diameter is 1 m^2/g, whereas it is 6 m^2/g for dusts 1 μm in diameter.

Testing showed that the SO_3 concentration in the converter gases is further influenced by gas temperature. When the temperature of the gases is less than 700°C at the converter outlet, SO_3 formation is inhibited and the concentration is maintained at a low, steady level of 0.1 to 0.2 percent SO_3 by volume. When the temperature of the gases is 800° to 1050°C at this point, SO_3

formation continues as the gas is cooled; recorded levels were 0.7 to 0.8 percent SO_3, and even as high as 1 percent SO_3 by volume.

THE CONVERTER GAS COOLING SYSTEM

A schematic representation of the particulate removal system for the converter gases is presented in Figure 5. Each converter is equipped with a gas-to-air heat exchanger, or gas cooler, and with a first-stage fan, from which the individual gas streams are mixed and cleaned in parallel-connected dry electrostatic precipitators (ESP's). The cleaned gases from the ESP's are again combined into a final collecting chamber and from there are picked up by second stage fans which deliver the gas to the sulfuric acid plant about 600 m from the converter aisle. Gas to the acid plant is still hot, 150° to 250°C.

Converter off-gases must be cooled to allow proper operation of the particulate control system. Direct cooling was the best answer at the Polish smelter. Air dilution was not a suitable approach because of the necessity of maintaining a sufficiently high SO_2 concentration for the acid plant. Water sprays unnecessarily increase gas volume, increase corrosion potential, and cause problems at the acid plant. Despite initial difficulties, the coolers developed by the Polish engineers have proved to be a practical method.

The principal design problems in a cooler are cracks because of thermal stress, dust accretion and corrosion caused by drops in temperature below the acid dew point, and structural burnout. The design problems are complicated by the temperature and gas flow fluctuations, as well as the high particulate loading and reactivity of the gas stream. Even with proper operation, gas fluctuations can sometimes cause temperature to drop below the acid dew point.

To minimize the effects of fluctuations in gas volume and temperature, the off-gas streams from several individual converters are combined in a mixing chamber after passing through the coolers. Use of this chamber ensures a relatively continuous gas flow to the ESP's at a temperature above the acid dew point (270°C). In addition, there is partial dust precipitation in the collector because of the reduction in gas velocity caused by the enlarged flow section. Dusts settle into hoppers, which are periodically emptied.

To evaluate the operation of the gas collecting system, detailed tests were conducted at the smaller smelter, which has 33-Mg converters. The gas removal system at this plant is

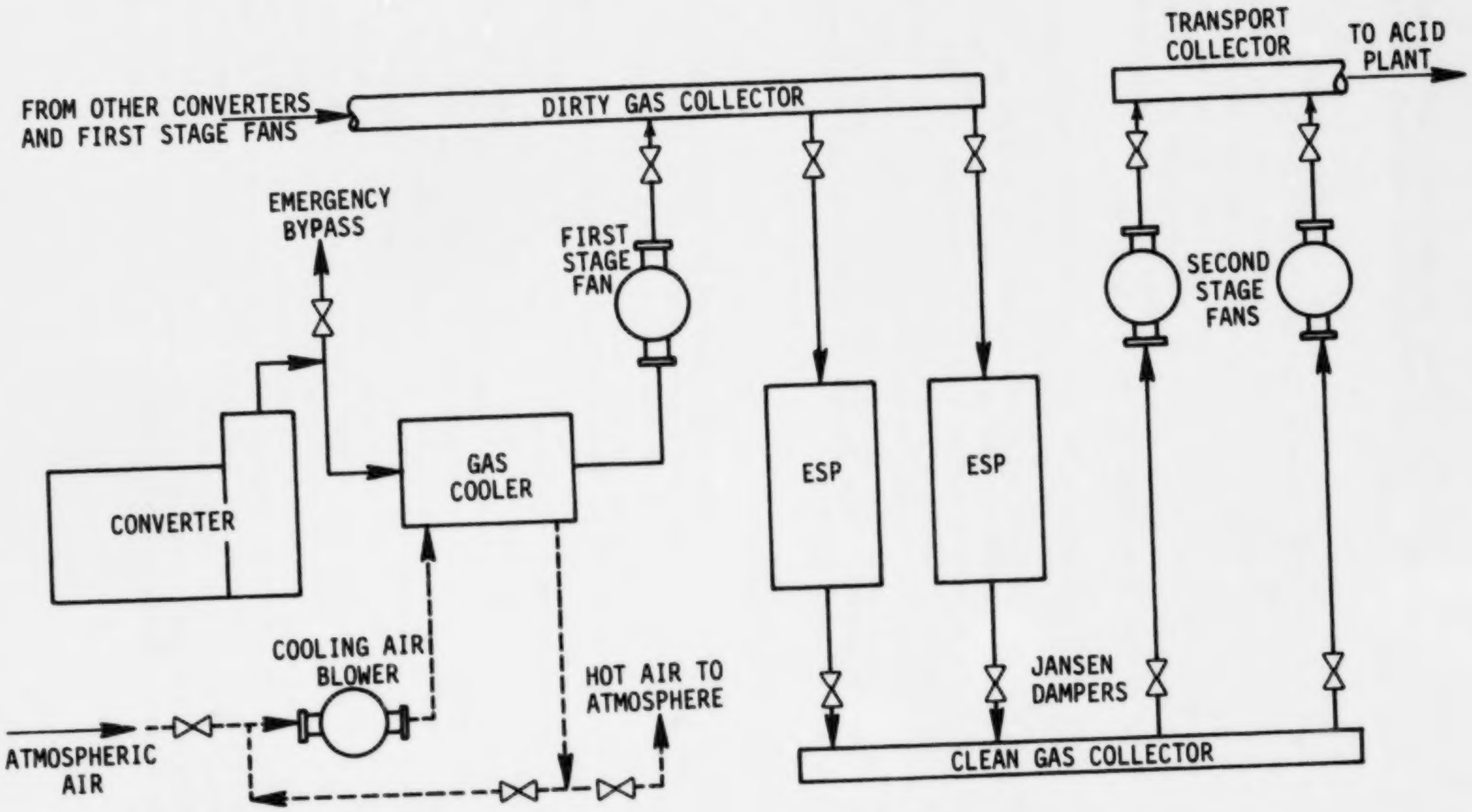

FROM OTHER CONVERTERS
AND FIRST STAGE FANS
DIRTY GAS COLLECTOR
TRANSPORT
COLLECTOR
TO ACID
PLANT
EMERGENCY
BYPASS
FIRST
STAGE
FAN
SECOND
STAGE
FANS
GAS
COOLER
CONVERTER
ESP
ESP
COOLING AIR
BLOWER
HOT AIR TO
ATMOSPHERE
JANSEN
DAMPERS
ATMOSPHERIC
AIR
CLEAN GAS COLLECTOR

Figure 5. Flow diagram - converter gas cleaning system.

identical to that used with the 80-Mg converters and similar test results would be expected. Three converters and two ESP's were operated during the test period.

Figure 6 charts the temperatures of the gas after it leaves the coolers and at the ESP inlets during part of the test period. The gas streams leaving the individual coolers are characterized by temperature fluctuations that reach as high as 300°C. These fluctuations, which result from changes in the blast at the converter, are most significant during the slag blow. The rise in temperature, which proceeds slowly, reaches the maximum value 0.5 to 1.0 hour after the initial blast begins.

During the test period, the gas temperature sometimes dropped below the acid dew point (270°C). This typically occurred for the first several minutes after the blast was put on a cold converter. Transferring these gases from the coolers directly to the ESP's would cause rapid corrosion of the electrodes and insulators because of sulfuric acid precipitation. The charts reproduced in Figure 6 demonstrate that the first mixing chamber stabilizes the gas temperature and protects the ESP's from these hazards. The range of temperature changes at the ESP's was less than 120°C.

The Original Cooler Design

The design of the gas coolers used with the Polish siphon converters has been modified over the years. The first design was an atmospheric natural-draft cooler, but it was soon replaced by the forced-air circulation cooler shown in Figure 7. Each side of this design had three double cooling sections, each of which consisted of two vertical cylindrical segments containing 14 pipes (355 mm diameter) attached to the upper and lower sieve bottoms. The first segments of the first section were attached to an entry chamber that received gas directly from the converter ductwork. The remaining five segments of each side of the cooler sat on a settling chamber that had two compartments. Each of the three sections was covered by a hood that changed the gas flow direction. Each of the vertical segments had two or three inlets that ensured proper draft and fresh air intake to the three sections. The cooler was constructed of standard carbon steel except for some elements of the first section, which were made of a special steel. The entry and settling chambers were lined with ceramic materials. The cooler weighed 400 Mg and was mounted on a special steel structure with several platforms at different levels to facilitate maintenance. The gas temperature was measured at the inlet; temperature and pressure were measured at the outlet.

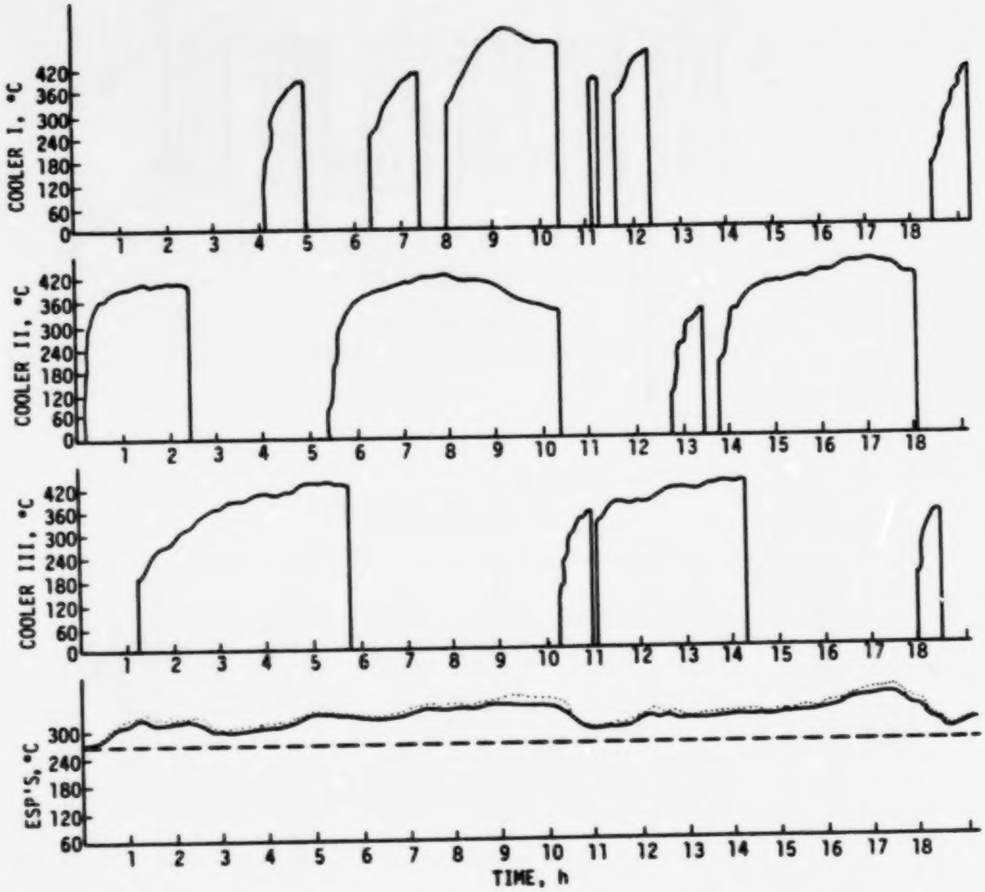


Figure 6. Converter gas temperature - cooler and collector outlets.

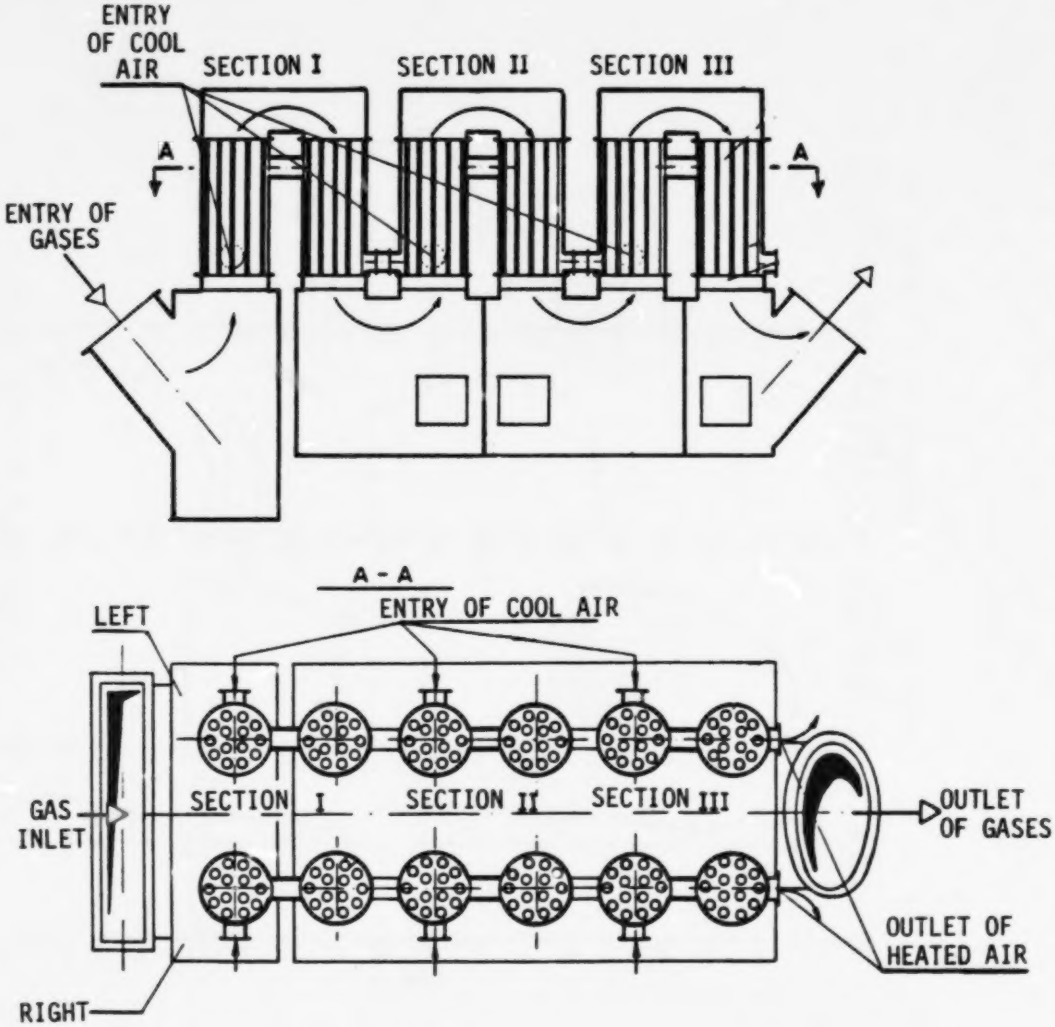

ENTRY OF COOL AIR
SECTION I
SECTION II
SECTION III
A
A
ENTRY OF GASES
A - A
ENTRY OF COOL AIR
LEFT
GAS INLET
SECTION I
SECTION II
SECTION III
OUTLET OF GASES
OUTLET OF HEATED AIR
RIGHT

Figure 7. Original forced-air circulation cooler.

The flow of converter gases and cooling air through the cooler is presented schematically in Figure 8. The gas was pulled through the ductwork by the first-stage fan to the entry chamber, which was an integral part of the cooler. A bypass to an emergency stack was located on the ductwork in front of the cooler inlet. The gases in the entry chamber were divided into two streams, which entered each side of the cooler. The gases were remixed in the settling chamber after passing through the first and second cooler section; they were also remixed in the exit chamber after passing through the final section. Precipitated dusts that accumulated in the settling chamber were removed periodically. Cooling air was forced into the interpipe space of the first vertical segment of each section. It flowed through the three sections parallel with the gas and was discharged to the atmosphere. Throttling-type control valves were used to regulate the air flow through each section and to ensure either partial or total heated air circulation. The operating parameters of the cooler are presented in Table 11.

TABLE 11. OPERATING PARAMETERS - ORIGINAL CONVERTER GAS COOLER

Cooling surface area	740 m^2
Inlet gas volume	35,000-40,000 Nm^3/h
Gas temperature:	
- Cooler inlet	850°-900°C
- First section outlet	600°-650°C
- Second section outlet	500°-420°C
- Cooler outlet	400°-450°C
Cooling air volume:	
- Nominal	20,000 Nm^3/h
- Total	36,000-40,000 Nm^3/h (avg)
Cooling air temperature:	
- First section outlet	180°-200°C
- Second section outlet	260°-280°C
- Third section outlet	300°-320°C

It was not possible to control the flow of cooling air within individual sections of the cooler, and local overheating occurred because of flow irregularities. This can be illustrated by referring to Figures 9 and 10, which are simplified representations of the temperatures of the gas, cooling air, and cooler walls in the three sections. Figure 9 represents two cases; in the first, air was only added to the first cooler section (Vp), whereas in the second, a lesser amount was added in the first (Vp-ΔVp), and additional air was added in each of the remaining two sections (ΔVp). The first case, in which all cooling air enters in the first section, was clearly advantageous; in the second case, wall temperatures could rise to above the temperature limit of the construction material. In addition,

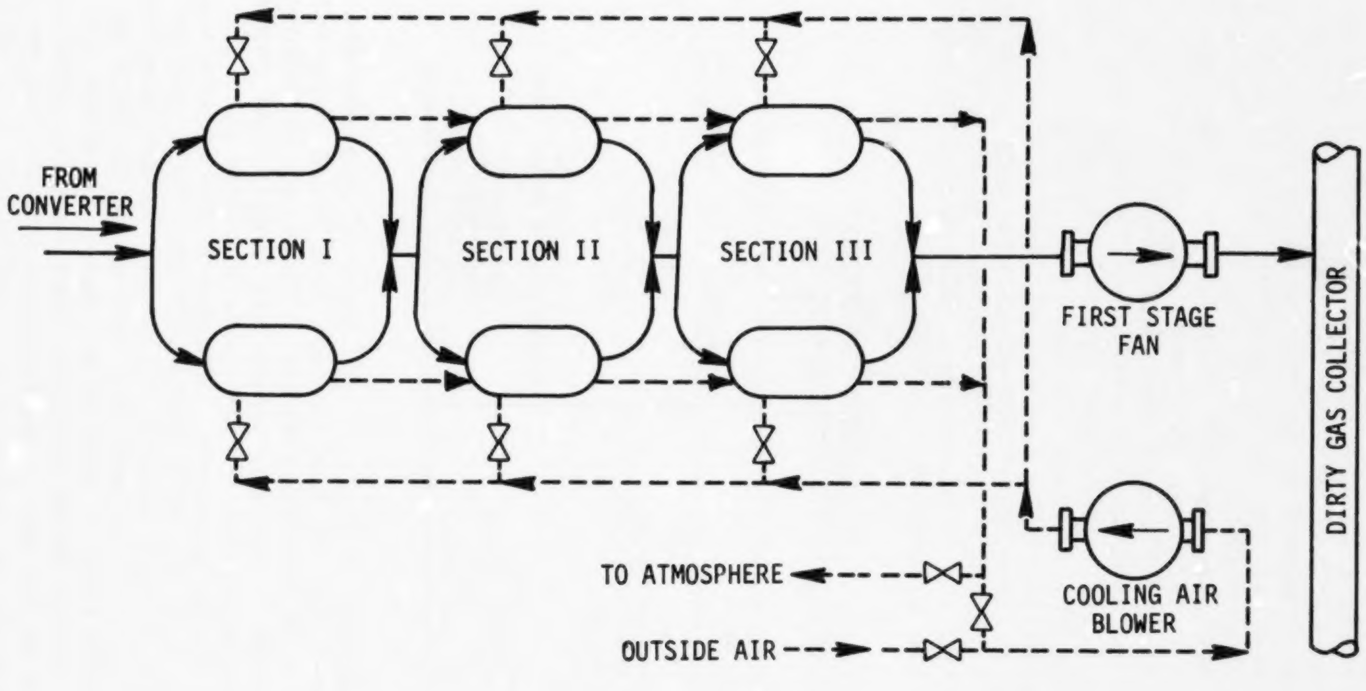

FROM CONVERTER
SECTION I
SECTION II
SECTION III
FIRST STAGE FAN
DIRTY GAS COLLECTOR
TO ATMOSPHERE
OUTSIDE AIR
COOLING AIR BLOWER

Figure 8. Flow of converter gases and cooling air through original cooler.

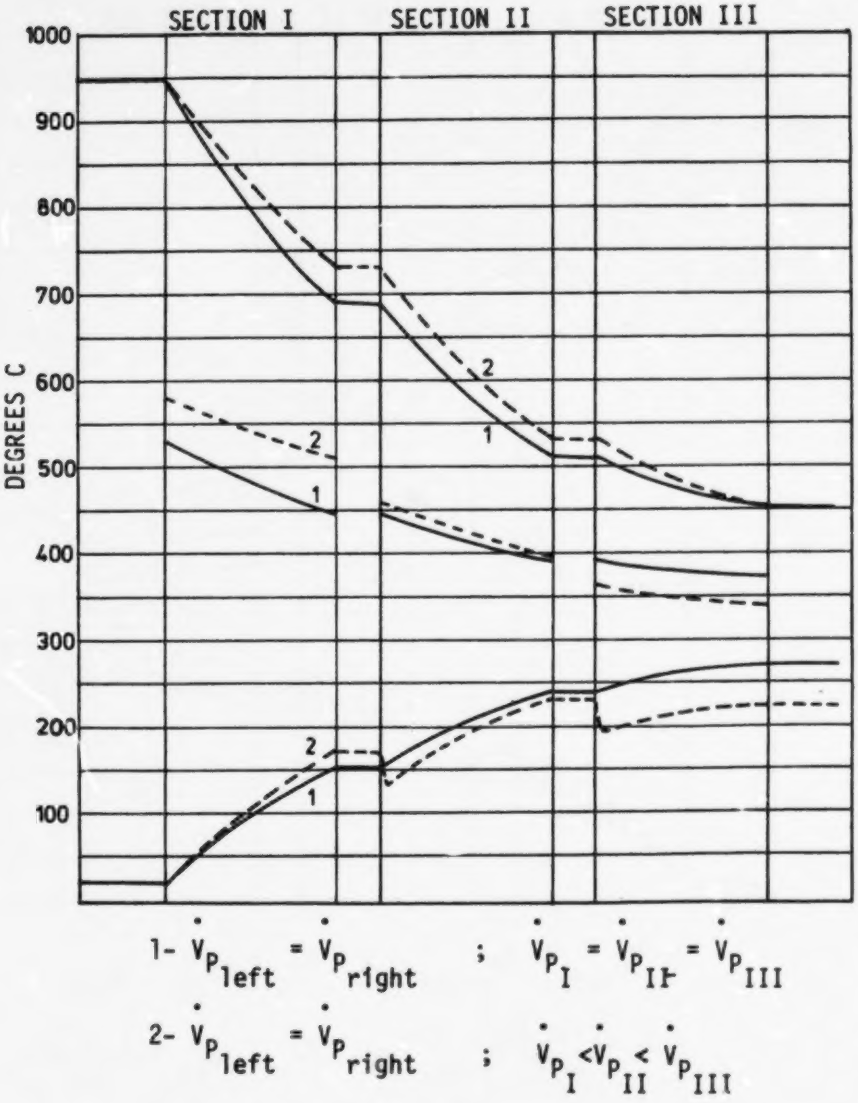
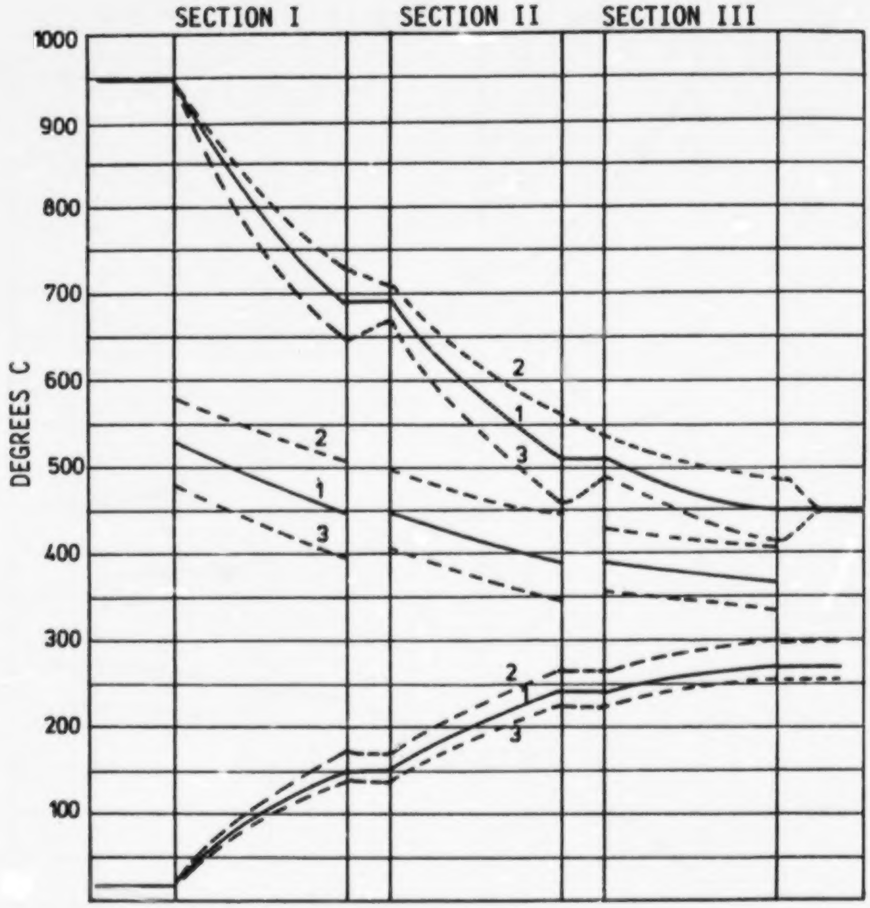


Figure 9. Cooler temperatures - symmetrical air flow.



1- $\dot{V}_{P_{left}} = \dot{V}_{P_{right}}$; $\dot{V}_{P_I} = \dot{V}_{P_{II}} = \dot{V}_{P_{III}}$

2-3- $\dot{V}_{P_{left}} < \dot{V}_{P_{right}}$; $\dot{V}_{P_I} = \dot{V}_{P_{II}} = \dot{V}_{P_{III}}$

Figure 10. Cooler temperatures - asymmetrical air flow.

less cooling air was required for the first case. Figure 10 reproduces the steady air flow curve from Figure 9, and adds two additional cases in which all of the flow entered the first section, but was unevenly distributed between the left and right sides. Total air flow was equal in both cases. These asymmetrical air flows also resulted in temperature irregularities. When less air was added to the left side, the temperature of the first section walls could have risen to a hazardous level.

The gases entering the cooler were laden with dusts from the converter, which settled on the walls of its internal ductwork and obstructed the gas flow. To evaluate the increase in resistance caused by these buildups, the pressure drop was monitored from the converter exhaust to the cooler outlet. Measurements were conducted for 30-minute periods just after the start of each operating cycle for 62 days (128 cycles). The precision of measurement was 5 mm H_2O. The arithmetic means of these resistances are presented in Table 12, and the points are plotted on Figure 11. This relationship can be described by the following formula: $\rho = A + Bi + Ci^2$. Using the test data, the parameters A, B, and C were calculated, resulting in the following equation:

$$\rho i = 135 + 0.0044i^2$$

in which: ρi = resistance in cooler and ductwork after cycle i

i = cumulative number of cycles from beginning of campaign

For the general case, the relationship would be as follows:

$$\rho i = \rho o + 0.0044i^2$$

in which ρo = resistance in cooler and ductwork at beginning of campaign.

A number of problems were experienced with this cooler design when it was operated with the 80-Mg converters. For each cooler section, it was not possible to control the irregularities in cooling air flow or to accurately predict the internal temperatures. Gradual dust accretion occurred in the ductwork, along with occasional acid corrosion of the structural elements in the third section. Most seriously, the first section overheated and subsequently burned through, causing the converter gases to be diluted with cooling air and the SO_2 concentration to drop seriously. The perforated bottoms of the tubular segments also cracked frequently because of thermal stress from the high-temperature gases.

TABLE 12. INCREASE IN COOLER RESISTANCE DURING CAMPAIGN

Cycle	Average resistance (mm H_2O)	Cycle	Average resistance (mm H_2O)	Cycle	Average resistance (mm H_2O)	Cycle	Average resistance (mm H_2O)
1	120	33	150	65	120	97	180
2	120	34	140	66	160	98	160
3	130	35	130	67	140	99	140
4	100	36	120	68	150	100	150
5	120	37	150	69	110	101	170
6	140	38	140	70	120	102	180
7	120	39	150	71	150	103	140
8	100	40	130	72	160	104	210
9	130	41	150	73	180	105	170
10	100	42	160	74	190	106	200
11	140	43	110	75	180	107	200
12	160	44	130	76	200	108	210
13	190	45	130	77	140	109	170
14	190	46	160	78	120	110	160
15	150	47	130	79	140	111	210
16	160	48	130	80	190	112	240
17	160	49	200	81	150	113	140
18	180	50	180	82	160	114	150
19	130	51	160	83	170	115	150
20	140	52	180	84	180	116	190
21	130	53	170	85	130	117	190
22	140	54	160	86	160	118	220
23	170	55	150	87	130	119	180
24	120	56	130	88	190	120	230
25	160	57	140	89	170	121	210
26	170	58	150	90	200	122	220
27	150	59	190	91	160	123	240
28	140	60	160	92	130	124	210
29	150	61	100	93	180	125	220
30	130	62	100	94	170	126	290
31	120	63	120	95	140	127	200
32	140	64	180	96	200	128	190

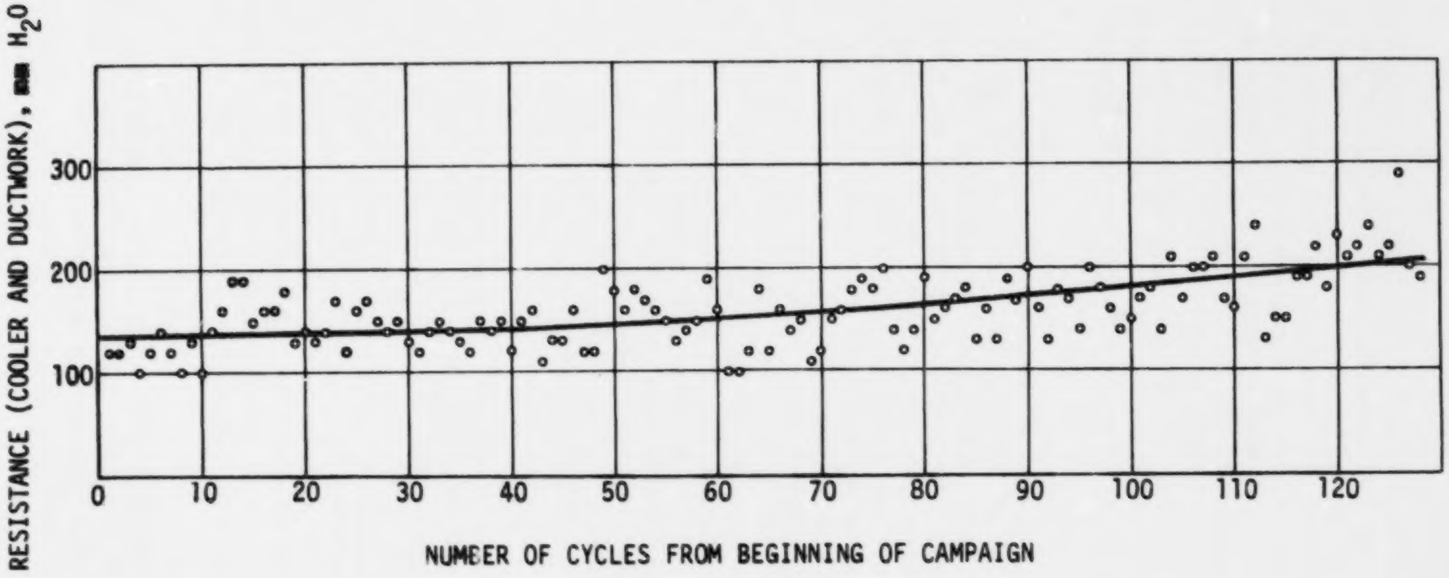


Figure 11. Increase in cooler resistance over time.

The Improved Cooler Design

Using the experience gained from these initial units, the coolers were replaced in 1977 by the new design shown in Figure 12. The most important modifications were the introduction of atmospheric cooling in the first section and a change in the direction of cooling air flow in the remaining sections. The change in the first section was accomplished simply by removing the internal tubular segments to allow the converter gases to flow through the heat-resistant steel jacket and radiate heat to the atmosphere. The general shape of the cooler was unchanged. The area of the cooling surface is 633 m^2, and the area reduction from the original design is due entirely to the changes in the first section.

A schematic of the flow of gases and cooling air through the new cooler is presented in Figure 13. As in the previous design, the gases are divided into two parallel streams as they enter the first cooler section and are remixed following each section. The vertical first-stage tubes, which are 1.75 m in diameter, protect the remaining sections from excessively high temperatures. The first-stage section has a total heat exchange area of 140 m^2, and the degree of its cooling effect depends primarily on the relative gas and ambient temperatures. Based on typical gas flow (32,000 Nm^3/h) and dry 10°C weather conditions, it has been calculated that the gases can be cooled from 950° to 841°C in the first stage. The second and third stages operate as countercurrent heat exchangers, with cooling air requirements which are greater than a comparable parallel cooler. The cooling air to each section is divided into four streams that feed each segment; the air is then discharged to the atmosphere. The total cooling air flow and the flow to each segment are adjusted with throttling-type control valves.

A test program was conducted to evaluate the new cooler design, and measurements were taken at the points shown in Figure 14. Tests were conducted over a period of a month during the middle of a converter's campaign. From 6 to 10 sets of measurements were recorded during an operating cycle, and the majority of parameters were recorded by continuous monitors. Some parameters were determined indirectly; for example, cooling air volume was calculated by differentiating the function of the mean dynamic and static pressures proportional to their midpoint values, allowing subsequent values to be measured at their midpoints only.

Based upon the measurements that were conducted, it was found that the gas flow through the cooler during copper blowing could be estimated by the following equation:

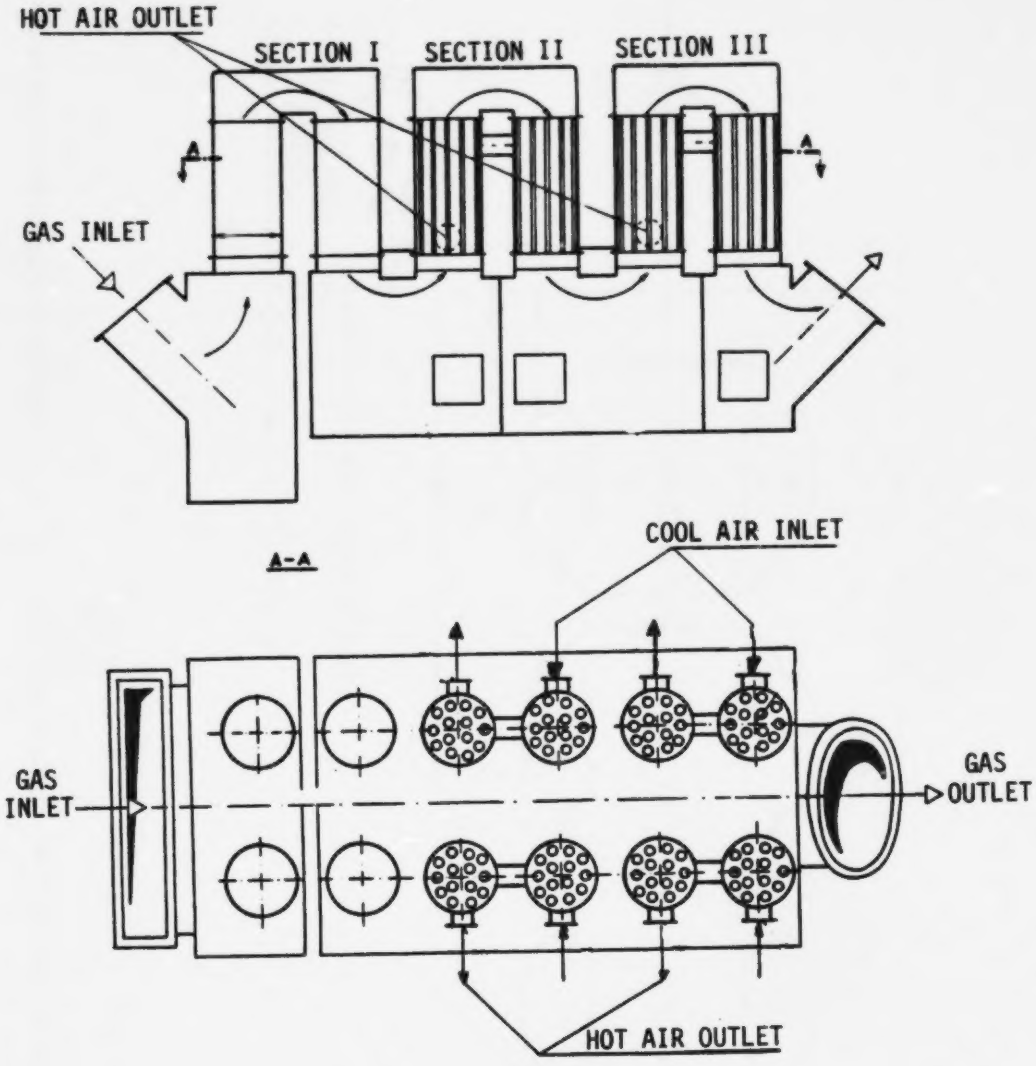

HOT AIR OUTLET
SECTION I
SECTION II
SECTION III
A
A
GAS INLET
A-A
COOL AIR INLET
GAS INLET
GAS OUTLET
HOT AIR OUTLET

Figure 12. Improved forced-air circulation cooler.

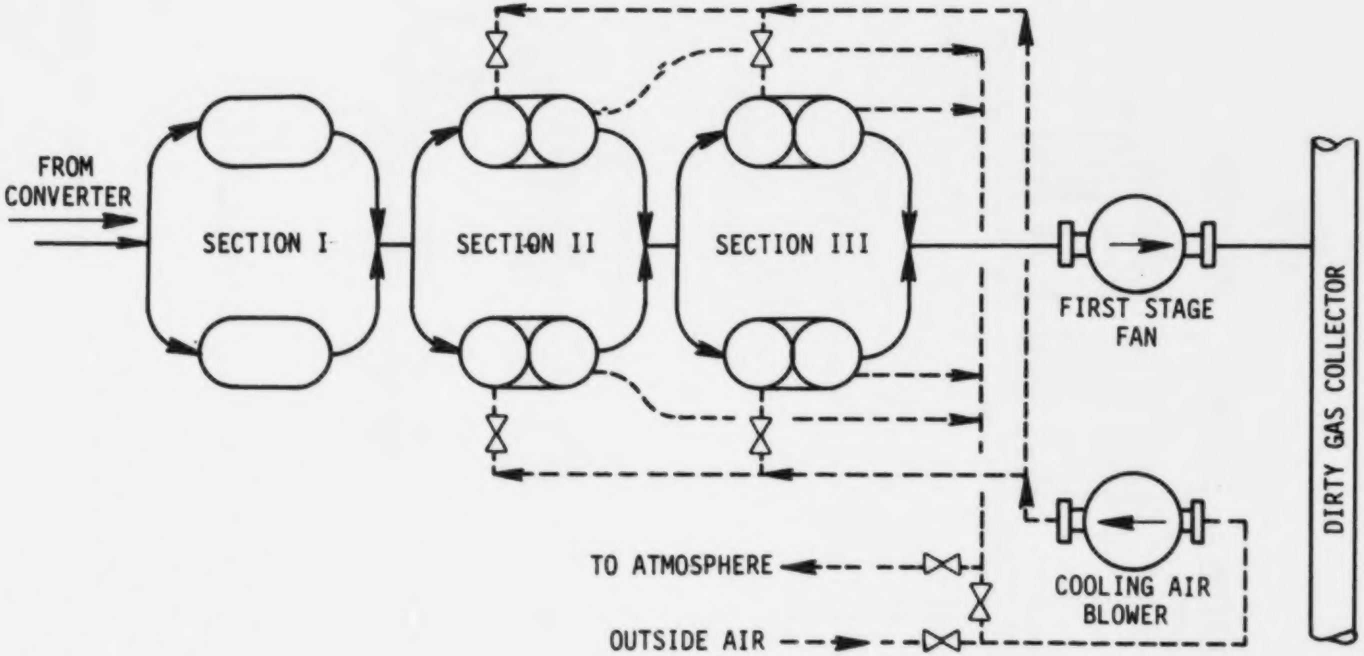

FROM
CONVERTER
SECTION I
SECTION II
SECTION III
FIRST STAGE
FAN
DIRTY GAS COLLECTOR
COOLING AIR
BLOWER
TO ATMOSPHERE
OUTSIDE AIR

Figure 13. Flow of converter gases and cooling air through improved cooler.

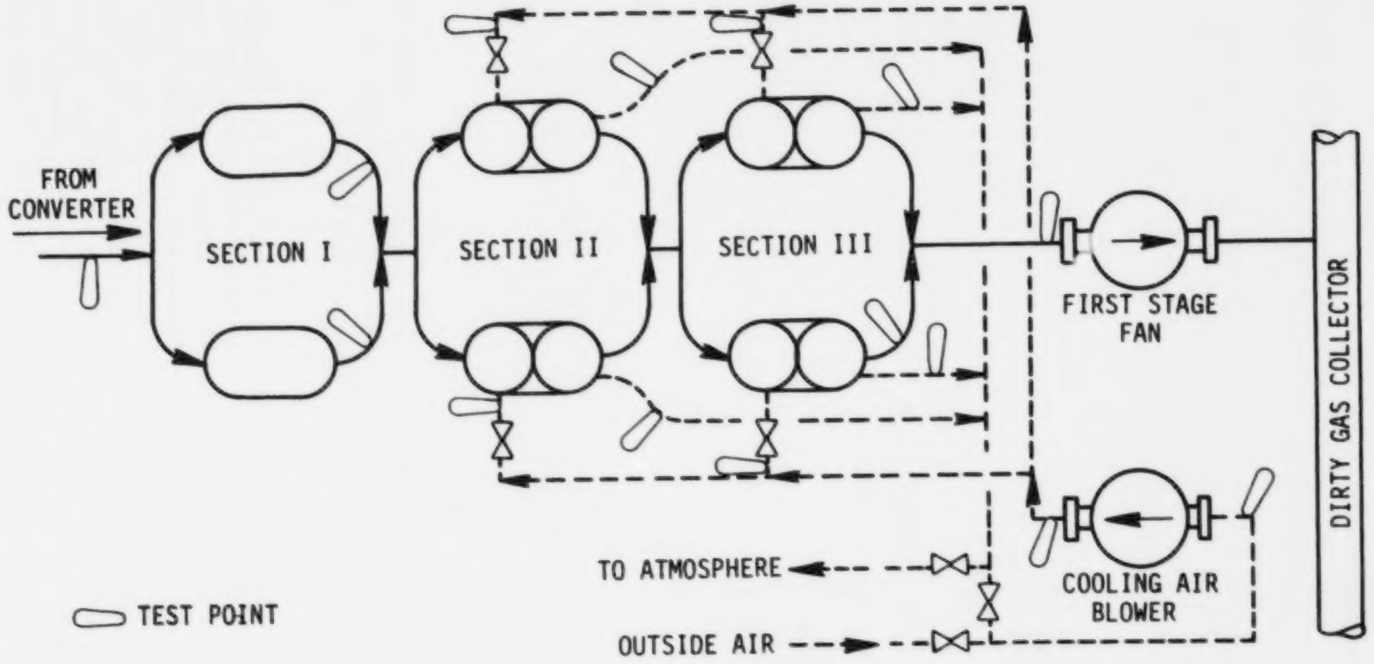

FROM
CONVERTER
SECTION I
SECTION II
SECTION III
FIRST STAGE
FAN
DIRTY GAS COLLECTOR
COOLING AIR
BLOWER
TO ATMOSPHERE
OUTSIDE AIR
TEST POINT

Figure 14. Measurement points for cooler test program.

$$Vg = \frac{18.6}{(SO_2)} \cdot Va \qquad \text{(Eq. 1)}$$

in which:
Vg = gas volume, Nm^3/h
SO_2 = average SO_2 content of converter gas, percent
Va = average blast intensity, Nm^3/h

Calculations were based on a copper content in the matte of 64 percent, and a degree of oxygen utilization of 0.885, as had been indicated by other tests. The amount of cooling air through the individual cooler sections was calculated by the following equation:

$$\frac{VAi}{Va} = \frac{\sqrt{\frac{\Delta pi}{Ti}}}{\sum_i \sqrt{\frac{\Delta pi}{Ti}}} \qquad \text{(Eq. 2)}$$

which results from the following assumption:

$$\Delta p \approx \rho \frac{W^2}{2} \qquad \text{(Eq. 3)}$$

in which: VA = air volume in given time, Nm^3/h;

$$VA \approx W \cdot \frac{\rho}{\rho u} = W \cdot \frac{Tu}{T} \qquad \text{(Eq. 4)}$$

i = number of individual cooler section
Δp = pressure drop across individual section, $\frac{N}{M^2}$
T = mean of air temperature at section inlet and outlet
U = conventional operating parameters
W = air rate
ρ = average air density

The stability of the coefficient of flow resistance results from the fact that the construction of the individual segments is identical, and therefore the operating conditions differ only slightly between sections.

Figure 15 is a plot of the gas and cooling air temperatures at various points in the cooler during two converter operating cycles. The blast flow intensity is also shown to illustrate the relationship between the converter operations and the cooler. During converter shutdown between cycles, the cooling air was shut off to avoid excessive temperature loss in the cooler. The cooling air was resumed about 15 to 30 minutes after the converter was rolled back to its vertical operating position, the exact timing determined by the temperature of the gases leaving the cooler.

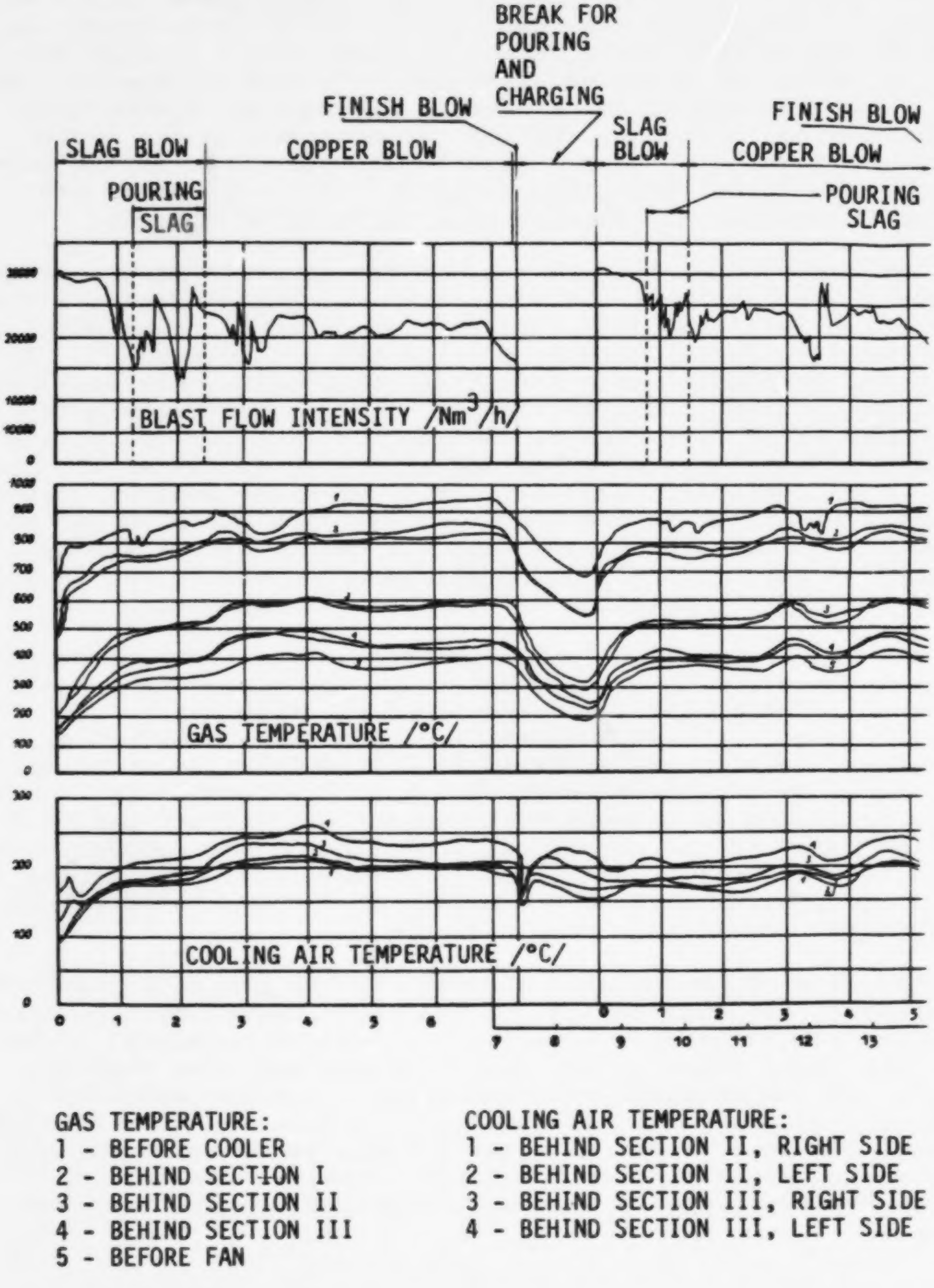


Figure 15. Gas and cooling air temperatures during two converter operating cycles.

Table 13 shows the average cooler operating parameters obtained over a period of 1 month operations. These tests were made during the copper blow part of the cycle, when conditions are more stable than during the slag blow, which is also too short to permit accurate measurements. Figure 15, however, shows that the cooler operating parameters are similar during both blowing periods. The air and gas temperatures in the cooler are plotted in Figure 16; these measurements were made at the inlets and outlets of the individual sections. The air curves are steeper because of the greater heat capacity of the gas.

The improved cooler remains hot for about 30 minutes after converter shutdown. If the temperature of the cooler is allowed to drop too much, the cool converter gas at the beginning of the next operating cycle could drop below the acid dew point. To reduce the problem of acid formation, the cooling air would have to be shut off until a sufficiently high gas temperature is reached at the cooler outlet. During the test period, delays of as much as 45 minutes were applied after the converter was rolled back to its operating position. This practice, however, may result in heat damage to the cooler walls. The use of a hot gas collecting chamber, from which hot air can be automatically distributed through the cooling air fans to retain heat in the cooler, offers a reasonable solution to this problem.

The degree of air dilution through valves and connectors varies inversely with the converter gas volume and the negative pressure in the cooler. The amount of dilution can be calculated by comparison of the SO_2 content in the gases at the cooler inlet and outlet, as reported in Table 13. This leakage, which represents about 21 percent of the total gas flow leaving the cooler, ranges from 2 to 40 percent. The amount of dilution provides a good indicator of the structural condition of the cooler, as any cracks or leaks will further reduce the SO_2 content. During the test period, which covered the middle of a converter's campaign, no signs of unusual wear were present.

The life of the tubular elements in the new cooler design is difficult to estimate because of the short operating experience since its installation. According to smelter personnel, damage after the first campaign was insignificant and less than expected. In the original cooler design, the first and third sections lasted 1 year, and the second 2 years. At least eight segments had to be repaired after a campaign by repairing cracks and changing pipes. After the first campaign with the new cooler, however, repairs were only required on the first two segments.

TABLE 13. AVERAGE COOLER OPERATING PARAMETERS

Inlet gas volume	23,000 Nm^3/h
Converter gas:	
- SO_2 content - cooler inlet	13.7%
- cooler outlet	11.3%
- Volume	31,500 Nm^3/h
- Air inleakage	6,700 Nm^3/h
- Temperature - cooler inlet	880°C
- first section outlet, left	812°C
- first section outlet, right	796°C
- second section outlet, left	593°C
- second section outlet, right	580°C
- final section outlet, left	473°C
- final section outlet, right	473°C
- cooler outlet	383°C
Cooling air:	
- Volume - inlet	43,000 Nm^3/h
- second section, left	12,000 Nm^3/h
- second section, right	14,400 Nm^3/h
- final section, left	8,900 Nm^3/h
- final section, right	7,700 Nm^3/h
- Temperature - inlet	18°C
- second section, left	224°C
- second section, right	213°C
- final section, left	236°C
- final section, right	232°C

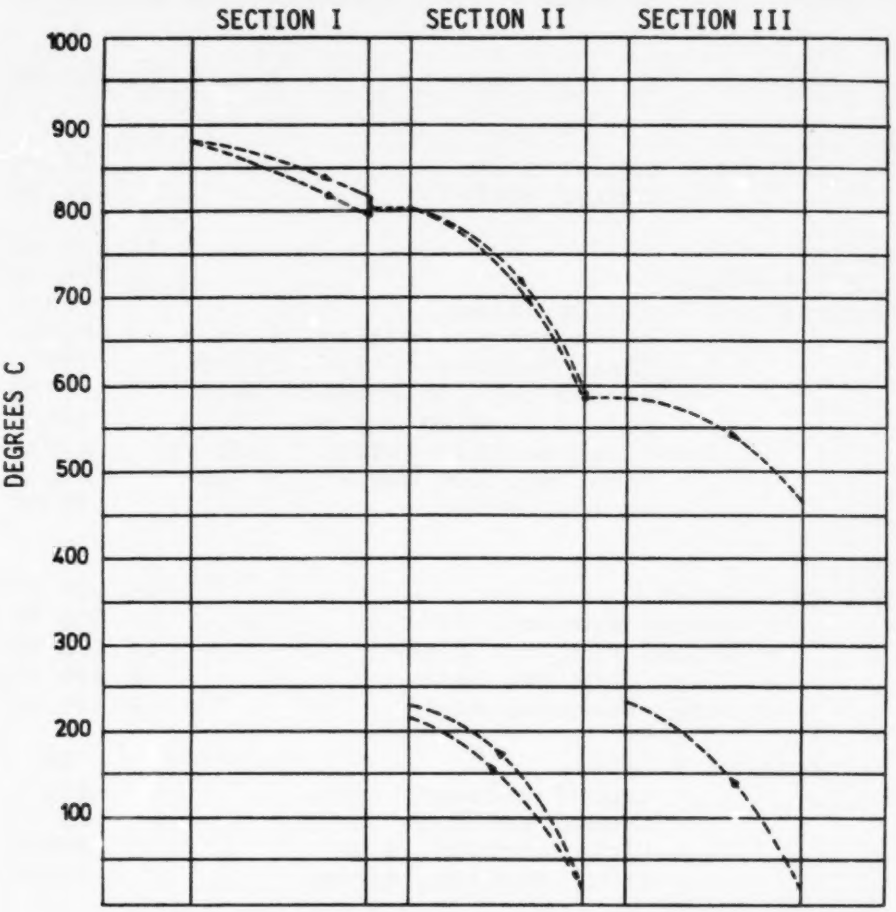


Figure 16. Gas and cooling air temperature.

The accumulation of dusts in the ductwork remains a problem even with the new cooler design. A layer of up to 10 mm thick that forms as dusts settle from the gases both reduces heat transfer and impedes gas flow. The dusts settle in layers that trap air in the interstices and further increase the insulative properties of the deposit. The settling rate depends primarily on the gas flow and physical properties of the dusts. The reduced cooling ability caused by the dust accumulation can be corrected by allowing additional dilution air to enter the cooler through flaps in the entry chamber. Since this would simultaneously reduce the SO_2 concentration, the preferred method of correcting the problem is periodic cleaning of the gas ducts. The tubular segments are sand-blasted between converter campaigns, as well as during a campaign if necessary. Sixteen to 24 hours are required for cleaning; the individual segments can be reached through their upper lids.

The only continuous monitoring equipment presently installed on the cooler system is a gas temperature indicator located in front of the first-stage fan. To more properly monitor the cooler's operation, the following measurements should also be taken:

Cooling air temperature at each segment outlet

Cooling air flow

Gas temperature at first section outlet

Gas flow resistance within the cooler

SO_2 concentration at cooler outlet

Ductwork and Fan Specifications

The off-gases from each converter are routed to its cooler through pipelines which are 2200 mm in diameter. Emergency stack bypasses, 1200 mm in diameter, are attached. The remaining ductwork from the cooler to the final collecting chamber is 1400 mm in diameter. All of this ductwork is made of acid-resistant steel, and all of the pipelines and equipment from the cooler outlet to the final collecting chamber are insulated with mineral wool enclosed in sheet aluminum. The gas mixing chambers are pipelines; the first has a diameter of 2008 mm, the second has a diameter of 1750 mm. Both are made of carbon steel lined with acid-proof brick. Dusts that precipitate in the first chamber are usually removed manually twice a year. The final collecting chamber, which has a diameter of 1966 mm, is also made of carbon steel lined with acid-proof brick. It is not insulated and is therefore equipped with a tank into which condensed sulfuric acid drains.

The first- and second-stage fans that pump the converter gases are identical. Both are made of acid-resistant steel, and they have rotors with blades shaped to prevent dust accretion. They are driven by direct-current electric motors to simplify speed control. Specifications of these fans and the cooling air fans are presented in Table 14.

TABLE 14. FAN SPECIFICATIONS

First- and second-stage converter gas fans:

Manufacturer - Sturtevant (Great Britain)
Model - 24/87D
Size - 135,000 m^3/h
Static pressure - 478 kg/m^2
Maximum speed - 1000 rpm
Power requirements - 300 kW

Cooling air fans:

Model - WPW 80/18
Size - 831,000 m^3/h
Static pressure - 260 kg/m^2
Speed - 985 rpm
Power requirements - 75 kW

ELECTROSTATIC PRECIPITATORS

The principal devices used for converter gas cleaning are horizontal, four-field dry ESP's constructed under license from Walther. They are made of acid-resistant steel to resist corrosion and are insulated with wool made of blown slag covered with sheet metal. A preheating system was constructed but it proved ineffective because of insufficient capacity. The construction of a new heater that will preheat the ESP to at least 270°C is projected. Dusts are removed from the hoppers once daily. Table 15 presents specifications of the ESP's used on the 80-Mg converters, and the design operating parameters.

The ESP's have performed satisfactorily when all operating requirements are met. From 95 to 99 percent of the particulates are removed with an inlet particulate loading of 5 to 13 g/Nm^3 (average 8 g/Nm^3), a gas temperature equal to or greater than 300°C, and a gas flow of 15,000 to 25,000 Nm^3/h. This lowers the outlet dust concentration to 0.15 to 0.30 g/Nm^3 (average 0.21 g/Nm^3). This cleaning efficiency is obtained with at least 50 kV applied to the electrodes and a corona discharge of 300 to 370 mA (for two fields). The outlet dust loading increases to 0.3 to

TABLE 15. ELECTROSTATIC PRECIPITATOR DATA

Model: H IV - 16/6 x 2/300 (Walther; Federal Republic of Germany)

Dimensions: 14,880 mm (l) x 3470 mm (w) x 11,420 mm (h); cross-sectional area - 28.8 m^2

Electrical field:	
Interpole distance	150 mm
Field dimensions	6.0 m (h) x 2.0 m (l)
Collectors	plate, sigma type
Dimensions	6.0 m (h) x 322 mm (w)
Number	6 x 17 rows
Total plate area	1576 m^2
Rapper control	Mechanically programed
Discharge electrodes	Strips with blades
Number per row	16
Total length	4250 m
Support	Quartz insulators
Rapper control	Continuous mechanical
Rapper frequency	0.4/min
Rapper drive motor	Electric; 0.6 kW; 750 rpm
Dust hoppers: 8 x 8.0 m^3	
Operating parameters	
Maximum gas volume	20,000 Nm^3/h
Gas velocity in field	0.45 m/s
Gas residence in field	18 s
Pressure drop	10 kg/m^2
Operating temperature:	
Maximum 400°C	
Minimum 270°C	
Maximum voltage applied	68 kV
Maximum corona current	375 mA

0.6 g/Nm^3 with a discharge of 200 to 300 mA, about 0.6 g/Nm^3 for 100 to 200 mA, and greater than 0.6 g/Nm^3 with a discharge below 100 mA.

Measurements were also conducted on the ESP's used on the 33-Mg converters, which are similar to those used at the larger smelter. Table 16 presents these results for gases averaging 5.7 g/Nm^3 at various corona discharge currents and gas velocities. A cleaning efficiency of about 97 percent is achieved during proper operating conditions. Since all the gas is further cleaned before release to the atmosphere, this efficiency is satisfactory. Table 17 through 20 present additional data relative to ESP performance under various operating conditions. The decrease in efficiency of particulate removal at temperatures below 290°C or at reduced corona discharge power is clearly illustrated. Figure 17 graphically depicts the relationship between ESP performance and gas velocity.

Dust Analysis

Because of problems encountered in direct sampling of the particulate content of the converter gases, the chemical composition of the dusts was determined on the basis of samples taken from the hoppers. Table 21 presents the results of these analyses, and analyses of dusts from the cooler hoppers. The content of free sulfuric acid particles was 1.72 to 4.96 percent in the dusts from the ESP hoppers, and 1.45 to 2.40 percent in those from the cooler hoppers. Dusts from the ESP hoppers at the larger smelter contained 0.93 to 4.14 percent free sulfuric acid. The high quantities of sulfate sulfur in the dusts suggest that the metallic compounds are principally in the sulfate form. This conclusion was confirmed by thermographic analysis of five dust samples from the ESP hoppers, which showed the following composition:

$PbSO_4$	36 to 63 percent
$ZnSO_4$	20 percent
$FeSO_4$	7 to 14 percent
$CuSO_4$	3 to 13 percent

Various trace metals are also found in the dusts, including 1.38 percent arsenic, 0.23 percent tin, and 0.05 percent bismuth. In addition, lesser quantities of antimony, silver, selenium, nickel, cadmium, and molybdenum were recorded. Arsenic occurs in the oxide form, and is especially important to control because of its toxic nature and its potential for poisoning the vanadium catalysts of the acid plant.

TABLE 16. ESP PERFORMANCE AS FUNCTION OF DISCHARGE POWER AND GAS VELOCITY

Current density (mA/m^2)	Corona discharge power (kW)	Gas velocity (m/s)	Power requirements (kW/1000 m^3)	Outlet particulate loading g/m^3	g/Nm^3
4 fields; temperature > 270°C					
0.028	1.94	0.282	0.066	0.174	0.385
0.075	5.88	0.132	0.430	0.093	0.195
0.105	9.01	0.242	0.359	0.110	0.225
0.179	16.60	0.167	0.961	0.062	0.132
0.176	16.22	0.229	0.682	0.066	0.140
0.282	26.74	0.145	1.773	0.041	0.090
0.286	27.49	0.219	1.210	0.061	0.132
0.267	27.51	0.285	0.933	0.095	0.205
0.332	32.59	0.133	2.354	0.044	0.095
0.347	35.68	0.304	1.131	0.079	0.170
0.417	44.80	0.148	2.928	0.042	0.087
0.417	44.40	0.378	1.132	0.058	0.123
2 fields; temperature > 270°C					
0.213	10.01	0.270	0.377	0.159	0.344

TABLE 17. ESP PERFORMANCE - TEMPERATURE $\geq$290°C POWER $\geq$15 KW

Operating parameters (kV/mA)		Corona discharge power (kW)			Gas volume (m^3/h)	Gas velocity (m/s)	Outlet particulate loading		Temperature (°C)
Fields 1/2	Fields 3/4	Fields 1/2	Fields 3/4	Total			(g/m^3)	(g/Nm^3)	
65/110	68/270	7.15	18.36	25.51	48,000	0.463	0.060	0.127	290
68/90	68/270	6.12	18.36	24.48	46,700	0.450	0.078	0.165	290
80/320	60/320	25.60	19.20	44.80	15,300	0.148	0.052	0.108	300
60/180	64/320	10.80	20.48	31.28	15,300	0.148	0.047	0.100	300
60/180	64/300	10.80	19.20	30.00	15,300	0.148	0.044	0.092	300
61/180	66/300	10.98	19.80	30.78	15,300	0.148	0.054	0.113	300
62/170	64/260	10.54	16.64	27.18	17,500	0.168	0.036	0.076	310
58/140	59/160	8.12	9.44	17.56	16,800	0.162	0.062	0.128	310
66/240	58/180	15.84	10.44	26.28	23,300	0.225	0.061	0.130	300
69/240	60/200	16.56	12.00	28.56	23,300	0.225	0.034	0.066	300
69/240	60/200	16.56	12.00	28.56	16,700	0.161	0.018	0.035	300
65/230	60/230	14.95	13.80	28.75	23,100	0.223	0.088	0.194	320
65/230	61/220	14.95	13.42	28.37	23,100	0.223	0.038	0.087	330
65/210	62/210	13.65	13.02	26.67	23,100	0.223	0.088	0.194	330
66/200	62/190	13.20	11.78	24.98	11,100	0.107	0.064	0.144	330
65/210	63/200	13.65	12.60	26.25	11,100	0.107	0.045	0.100	330
70/280	64/200	19.60	12.80	32.40	28,300	0.273	0.069	0.147	300
72/290	63/230	20.88	14.49	35.37	28,300	0.273	0.065	0.138	300
72/280	64/220	20.16	14.08	34.24	29,500	0.285	0.085	0.185	300
63/200	71/210	12.60	14.91	27.51	29,500	0.285	0.095	0.205	300
66/250	62/300	16.50	18.60	35.10	34,400	0.332	0.088	0.192	315
68/260	66/310	17.68	20.46	38.14	34,400	0.332	0.080	0.176	315
68/270	66/310	18.36	20.46	38.82	34,400	0.332	0.083	0.182	315
63/110	60/240	6.93	15.12	22.05	20,400	0.197	0.057	0.130	340
74/340	63/340	25.16	21.42	46.58	35,700	0.344	0.041	0.090	310
73/320	65/325	24.28	21.13	45.41	35,700	0.344	0.051	0.110	300
74/300	61/330	22.20	20.13	42.33	35,700	0.344	0.055	0.119	305
74/290	74/320	21.46	23.68	45.14	43,000	0.415	0.060	0.127	300
74/290	63/320	21.46	20.16	41.62	43,000	0.415	0.059	0.130	310
74/330	63/330	24.42	20.79	45.21	31,150	0.300	0.048	0.101	290
74/310	61/320	22.94	19.52	42.46	44,800	0.432	0.062	0.130	290
63/150	57/100	9.45	5.70	15.15	22,000	0.212	0.064	0.143	330
63/150	55/100	9.45	5.50	14.95	22,000	0.212	0.066	0.150	330
62/155	56/100	9.61	5.60	15.21	24,700	0.238	0.101	0.220	305
62/160	57/100	9.92	5.70	15.62	24,700	0.238	0.057	0.121	300

(continued)

TABLE 17 (continued).

Operating parameters (kV/mA)		Corona discharge power (kW)			Gas volume (m^3/h)	Gas velocity (m/s)	Outlet particulate loading		Temperature (°C)
Fields 1/2	Fields 3/4	Fields 1/2	Fields 3/4	Total			(g/m^3)	(g/Nm^3)	
61/220	62/280	13.42	17.36	30.78	14,700	0.141	0.035	0.077	325
62/200	60/280	12.40	16.80	29.20	11,800	0.113	0.023	0.052	320
57/290	57/200	16.53	11.40	27.93	11,800	0.113	0.031	0.069	330
65/250	67/225	16.25	17.08	33.33	10,500	0.101	0.024	0.051	305
66/260	65/270	17.16	17.55	34.71	14,100	0.136	0.024	0.051	305
63/290	69/270	18.27	18.63	36.90	15,900	0.153	0.076	0.168	315
65/280	58/280	18.20	16.24	34.44	13,900	0.134	0.050	0.110	310
64/250	58/260	16.00	15.08	31.08	9,600	0.092	0.021	0.097	315
64/210	60/260	13.44	15.60	29.04	9,600	0.092	0.018	0.039	310
60/200	61/240	12.00	14.64	26.64	21,800	0.210	0.060	0.129	300
62/100	57/170	6.20	9.69	15.89	23,400	0.226	0.064	0.139	290
64/95	58/165	6.08	9.57	15.65	23,400	0.226	0.062	0.131	290
60/125	59/135	7.50	7.96	15.46	18,000	0.174	0.059	0.126	300
58/100	60/170	5.80	10.20	16.00	20,000	0.193	0.061	0.135	320
57/100	60/180	5.70	10.80	16.80	20,000	0.193	0.048	0.106	320
57/100	60/180	5.70	10.80	16.50	22,000	0.212	0.052	0.115	320
64/100	61/180	6.40	10.98	17.38	17,500	0.169	0.072	0.153	295
60/120	60/280	7.20	16.80	24.00	20,600	0.199	0.064	0.136	295
60/125	58/300	7.50	17.40	24.90	20,000	0.193	0.056	0.119	290
Mean	-	-	-	28.88	-	0.226	0.057	0.122	-

TABLE 18. ESP PERFORMANCE - TEMPERATURE $\geq$290°C, POWFR <15 KW

Operating parameters (kV/mA)		Corona discharge power (kW)			Gas volume (m^3/h)	Gas velocity (m/s)	Outlet particulate loading		Temperature (°C)
Fields 1/2	Fields 3/4	Fields 1/2	Fields 3/4	Total			(g/m^3)	(g/Nm^3)	
51/50	48/60	2.55	2.88	5.43	16,800	0.162	0.102	0.211	310
50/50	50/45	2.50	2.25	4.75	16,800	0.162	0.099	0.205	310
-	61/200	-	12.20	12.20	18,800	0.181	0.108	0.236	310
-	64/220	-	14.08	14.08	18,800	0.181	0.081	0.179	320
45/20	46/20	0.90	0.92	1.82	28,000	0.270	0.075	0.160	325
43/20	44/20	0.86	0.88	1.74	27,700	0.267	0.122	0.278	340
45/20	45/20	0.90	0.90	1.80	27,700	0.267	0.212	0.470	325
44/20	44/20	0.88	0.88	1.76	27,700	0.260	0.167	0.380	340
48/50	48/50	2.40	2.40	4.80	12,600	0.122	0.078	0.165	295
-	58/120	-	6.96	6.96	26,800	0.258	0.114	0.244	300
53/20	46/30	1.06	1.38	2.44	31,100	0.300	0.184	0.405	320
50/20	43/25	1.00	1.08	2.08	34,100	0.329	0.283	0.618	315
-	55/100	-	5.50	5.50	39,200	0.378	0.300	0.672	325
65/115	55/100	7.48	5.50	12.98	18,800	0.181	0.039	0.096	330
58/70	52/70	4.06	3.64	7.70	13,900	0.134	0.036	0.080	315
57/70	50/70	3.99	3.50	7.49	13,900	0.134	0.041	0.098	315
58/50	50/50	2.90	2.50	5.40	13,700	0.132	0.022	0.048	315
58/50	52/80	2.90	4.16	7.06	13,700	0.132	0.078	0.169	305
55/50	50/70	2.75	3.50	6.25	13,700	0.132	0.087	0.186	300
55/50	51/70	2.75	3.57	6.32	10,000	0.096	0.064	0.137	300
50/50	51/70	2.50	3.57	6.07	10,000	0.096	0.142	0.300	290
-	60/240	-	14.40	14.40	15,900	0.153	0.134	0.296	315
60/200	-	12.00	-	12.00	31,500	0.384	0.254	0.550	305
60/120	58/130	7.20	7.54	14.74	24,000	0.231	0.056	0.119	300
59/120	59/120	7.08	7.08	14.16	18,000	0.174	0.063	0.135	300
Mean	-	-	-	7.20	-	0.205	0.118	0.257	-

TABLE 19. ESP PERFORMANCE - TEMPERATURE 270°-290°C

Operating parameters (kV/mA)		Corona discharge power (kW)			Gas volume (m^3/h)	Gas velocity (m/s)	Outlet particulate loading		Temperature (°C)
Fields 1/2	Fields 3/4	Fields 1/2	Fields 3/4	Total			(g/m^3)	(g/Nm^3)	
53/90	64/215	4.77	13.76	18.53	26,800	0.258	0.090	0.186	280
54/90	64/220	4.86	14.08	18.94	26,800	0.258	0.075	0.155	280
53/90	64/220	4.77	14.08	18.85	12,000	0.116	0.071	0.143	270
76/320	65/340	24.32	22.10	46.42	44,800	0.432	0.085	0.180	285
50/70	48/60	3.50	2.88	6.38	15,800	0.152	0.090	0.185	280
55/40	56/120	2.20	6.72	8.92	23,700	0.228	0.115	0.232	270
55/45	55/115	2.48	6.33	8.81	24,800	0.239	0.064	0.133	285
56/40	56/120	2.24	6.72	8.96	24,800	0.239	0.094	0.192	280
54/45	56/120	2.43	6.72	9.15	26,800	0.258	0.121	0.250	285
61/200	61/245	12.20	14.95	27.15	21,300	0.205	0.060	0.125	280
64/100	58/170	6.40	9.86	16.26	22,700	0.219	0.071	0.148	285
-	64/180	-	11.52	11.52	23,100	0.223	0.092	0.185	275
-	66/180	-	11.88	11.88	20,000	0.193	0.086	0.176	275
60/100	66/160	6.00	10.56	16.56	15,900	0.153	0.064	0.131	275
Mean	-	-	-	16.31	-	0.227	0.084	0.173	-

TABLE 20. ESP PERFORMANCE - TEMPERATURE <270°C

Operating parameters (kV/mA)		Corona discharge power (kW)			Gas volume (m^3/h)	Gas velocity (m/s)	Outlet particulate loading		Temperature (°C)
Fields 1/2	Fields 3/4	Fields 1/2	Fields 3/4	Total			(g/m^3)	(g/Nm^3)	
54/100	55/90	5.40	4.95	10.35	19,500	0.188	0.169	0.335	260
53/100	52/100	5.83	5.20	11.03	19,500	0.188	0.262	0.520	260
51/90	49/80	4.59	3.92	8.51	15,400	0.148	0.263	0.528	260
53/90	50/80	4.77	4.00	8.77	12,600	0.122	0.150	0.307	260
-	50/60	-	3.00	3.00	15,700	0.151	0.262	0.538	260
54/85	48/60	4.59	2.88	7.47	15,700	0.151	0.197	0.405	260
55/60	64/215	3.30	13.76	17.06	14,200	0.137	0.134	0.280	265
-	63/280	-	17.64	17.64	14,800	0.143	0.228	0.415	255
-	64/290	-	18.56	18.56	14,000	0.135	0.212	0.386	255
-	63/290	-	18.27	18.27	14,000	0.135	0.294	0.547	265
-	60/210	-	12.60	12.60	10,800	0.104	0.097	0.193	260
60/160	64/210	9.60	13.44	23.04	10,800	0.104	0.065	0.129	260
62/160	64/220	9.92	14.08	24.00	10,800	0.104	0.068	0.136	260
60/160	64/210	9.60	13.44	23.04	9,800	0.095	0.063	0.127	265
-	46/100	-	4.60	4.60	16,600	0.160	0.234	0.418	265
Mean	-	-	-	13.86	-	0.138	0.180	0.351	-
-	57/130	-	7.41	7.41	40,700	0.393	0.870	0.740	245
72/130	59/210	9.36	12.39	21.75	29,700	0.286	1.167	2.150	220
74/90	59/210	6.66	12.39	19.05	29,700	0.286	0.973	1.810	225
74/80	59/200	5.92	11.80	17.72	29,000	0.280	1.178	2.280	245
60/70	60/200	4.20	12.00	16.20	42,000	0.405	1.174	2.250	240
Mean	-	-	-	16.43	-	0.330	1.072	2.046	-

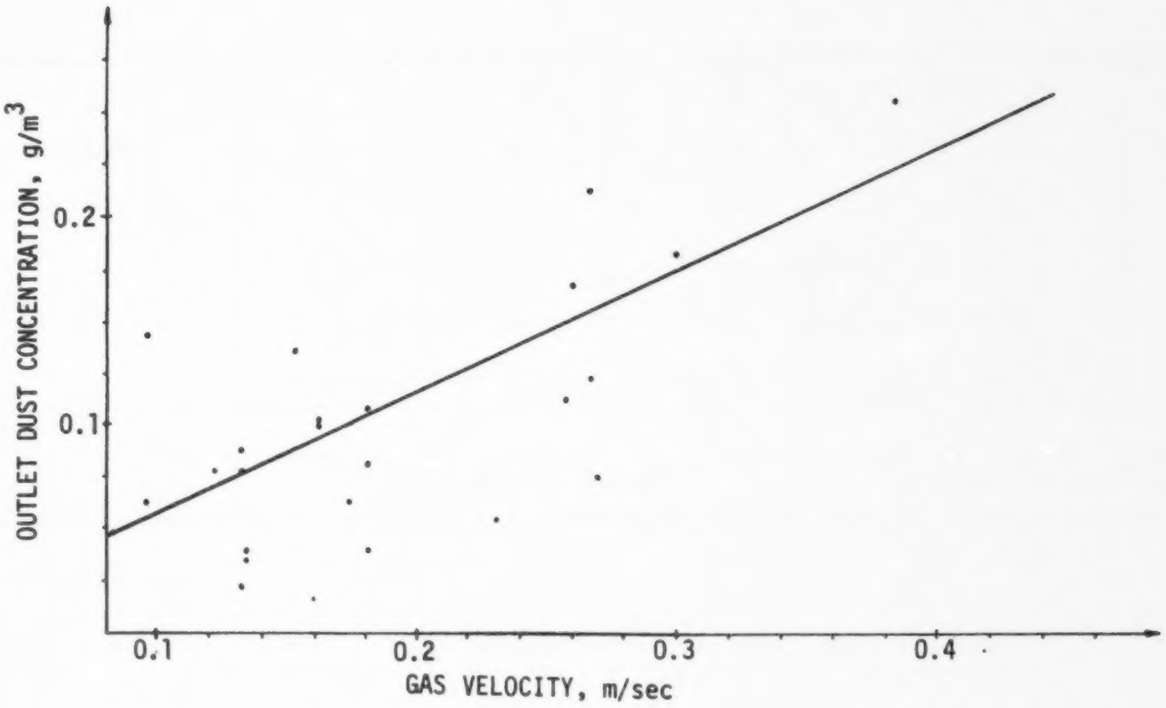


Figure 17. ESP performance as function of gas velocity.

TABLE 21. ANALYSES OF DUSTS FROM ESP AND COOLER HOPPERS

Species	ESP hopper dusts		Cooler hopper dusts	
	Range (24 tests) (%)	Average (%)	Range (4 tests) (%)	Average (%)
Lead	44.66-55.10	48.75	33.73-47.90	41.28
Copper	0.38-1.78	0.82	2.25-8.80	5.70
Zinc	4.70-7.66	6.39	2.70-5.33	4.16
Iron	0.05-0.20	0.10	1.20-5.20	2.58
Total sulfur	11.69-12.97	12.25	11.34-14.60	12.99
Sulfate sulfur	11.50-12.38	12.04	10.86-14.10	12.58

The physical properties of the converter dusts have a direct effect on particulate removal, collection, and transport. These characteristics were determined based upon samples taken from the ESP's on the 33-Mg converters; only verification tests were conducted at the larger smelter because of the similarities in dust properties. The presence of free sulfuric acid particles that wet the dusts influences their physical properties, as shown in Table 22. The great differences between the bulk and true physical densities evident from these data are indicative of the highly uneven surface of the individual dust grains.

The angle of repose of the dusts collected in the ESP is shown in Figure 18. The increase in this angle in the direction of the gas flow is characteristic. The largest quantities of dusts precipitate in the first fields. The subsequent decrease in grain loading favors the formation of sulfuric acid particles that wet the dusts, causing them to adhere to the hopper walls. This phenomenon was confirmed by analyses of the free sulfuric acid particle content in the following samples taken from the individual hoppers.

Field 1	0.93 percent H_2SO_4
Field 2	2.86 percent H_2SO_4
Field 3	2.94 percent H_2SO_4
Field 4	4.14 percent H_2SO_4

The electrostatic properties of the dusts, in particular resistivity, affect particulate removal in the ESP's. Because of the lack of equipment for field testing, dust samples were heated to 320° to 360°C in the laboratory (which is equivalent to temperatures in the ESP), and resistivity was measured as the dusts cooled. The results are shown in Table 23. The measured resistivities were suitable for particulate removal in an ESP for temperatures equal to or greater than 280°C. When the gas temperature falls below the acid dew point, however, acid particles precipitate on the dust grains and cause a severe drop in resistivity. In such cases, the dusts become overcharged upon contact with the settling plate and are returned to the gas stream. This is the principal reason for the drop in cleaning efficiency at temperatures below the acid dew point.

Fractional particle size analyses of the converter dusts were conducted on samples taken from the hoppers, also because of difficulties in sampling the gas stream directly. The results are presented in Table 24. These values cannot be directly related to the dusts present in the gas itself, however, because of grain coagulation after settling.

TABLE 22. DENSITY OF CONVERTER DUSTS

Density (g/cm^3)	33-Mg converter 1% H_2SO_4 in dust			33-Mg converter 3-4% H_2SO_4 in dust			80-Mg converter 1.5% H_2SO_4 in dust
	Avg	Min	Max	Avg	Min	Max	Avg
Bulk density	0.44	0.32	0.61	0.97	0.65	1.11	0.58
Shaken density	0.53	0.38	0.74	1.18	0.75	1.38	0.69
True physical density	4.30	3.72	4.84	4.28	3.60	4.73	4.12

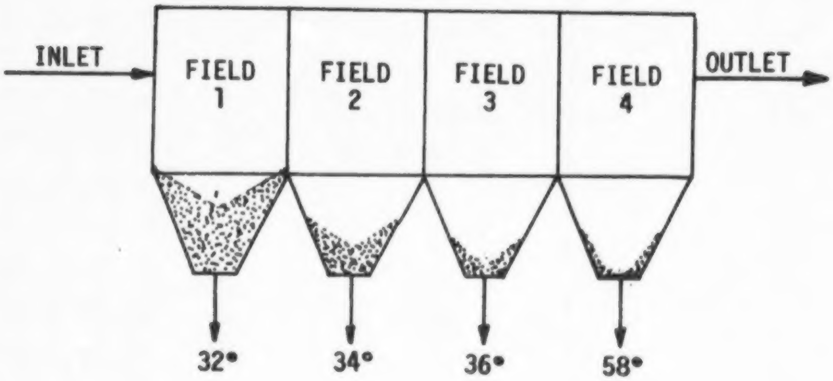

INLET
FIELD 1
FIELD 2
FIELD 3
FIELD 4
OUTLET
32°
34°
36°
58°

Figure 18. Angle of repose of dusts in ESP hoppers.

TABLE 23. CONVERTER DUST RESISTIVITY

Temperature (°C)	Resistivity (ohm cm)	
	Sample 1	Sample 2
375	-	2.70×10^6
340	-	3.12×10^6
330	-	3.43×10^6
320	0.10×10^6	4.37×10^6
300	0.20×10^6	4.58×10^6
280	0.25×10^6	6.25×10^6
260	0.27×10^6	9.34×10^6
250	-	1.35×10^7
230	0.59×10^6	2.19×10^7
210	1.06×10^6	3.96×10^7
195	-	7.60×10^7
190	2.11×10^6	-
180	-	1.35×10^8
170	5.28×10^6	1.87×10^8
160	-	3.12×10^8
150	1.05×10^7	4.79×10^8
140	-	7.28×10^8
130	2.11×10^7	1.25×10^9
115	-	2.40×10^9
105	5.28×10^8	-
100	-	5.57×10^9
90	2.95×10^9	-
85	-	1.46×10^{10}
80	8.45×10^9	-
60	2.53×10^{10}	8.85×10^{10}

TABLE 24. CONVERTER DUST GRAIN SIZE

Grain size (μm)	Fraction (%)
>60	24.62
40-60	12.85
30-40	9.80
20-30	9.30
10-20	13.21
5-10	11.32
0-5	18.90

Operating Problems

Frequent damage to the insulators is the most severe problem encountered with the ESP's because servicing requires that the power be cut off to two fields. This results in a decrease in particulate removal efficiency and therefore an increased dust content in the gases supplied to the acid plant. This naturally increases operating costs. The insulators operate in more severe conditions than any other known application, including fluctuating thermal and mechanical stresses and chemical reactivity of the gases. During the first 6 months of operation of the ESP's on the 33-Mg converters, 21 bushings, 19 support insulators, and 38 drive insulators were replaced. Although specific data are not available, similar wear is likely on the ESP's on the larger converters. The principal reason for insulator wear is the continuous leakage of current from their surface to the dust and acid. The problem may be mitigated by maintaining the operating temperature above the acid dew point and the pressure between 0 to 10 mm H_2O. This is not always possible in practice, however, and washing the insulators with hot, clean gas is necessary.

A second problem is the formation and precipitation of sulfuric acid, which can occur in the cooler, ESP, wet scrubbers, and ductwork. Sulfuric acid causes corrosion throughout the gas cleaning system. Precipitated dusts settle out in the ductwork and reduce heat exchange. Acid precipitation also reduces the ESP's dust removal efficiency, damages insulators, and creates problems with dust removal and transport from the hoppers. Acid precipitation occurs when the partial pressure of the acid and water vapor in the gas exceeds the saturation pressure. The water vapor comes from either the blast air or inleakage into the system, and it is assumed that the absolute humidity of the converter gases is equivalent to that of the ambient air (3 to 16 g

H_2O/Nm^3, average 8 g H_2O/Nm^3). Acid precipitates mainly on the internal surfaces of the ESP, which are cooler than the gas. The acid dew point can be lowered by decreasing the gas humidity through dehumidification of the converter blast air. This would increase operating costs, however, and would be only a partial solution to the problem, as moist air would still enter the system through the converter mouth and through leaks.

The acid dew point can also be lowered by a decrease in the SO_3 content, which can be effected by either a chemical binding of the SO_3 or a rapid cooling of the gas to about 700°C. The results of lime addition to tie up SO_3 are presented in Appendix A. Rapid cooling may be obtained by either water injection or air dilution. Water injection has not been adopted in Polish smelters. To cool the gases from 1250° to 700°C would require evaporation of 0.2 kg of water per Nm^3 of gas, which is about 5 m^3 H_2O/h at a blast intensity of 25,000 Nm^3/h. Since this approach increases gas humidity, poor cooling results, also a drop in acid dew point of only 20° to 25°C. Allowing 80 percent dilution through the converter mouth will lower the dew point by 40°C, but this approach cannot be recommended because of the possibility of cooling the converter and the reduction in SO_2 concentration. Thus, the only appropriate method of reducing acid precipitation in the ESP is by maintaining the gas temperature above 300°C.

FUGITIVE EMISSIONS

During slag blowing of the converter, many reactions occur, including the oxidation of FeS with a simultaneous evolution of SO_2 into the furnace interior over the molten bath. A negative pressure at the gas offtake must be maintained to prevent fugitive discharges through the converter mouth during this period. Each of the several slag pours, during which the converter is rolled forward and the blast is reduced in intensity, results in a pressure pulsation in the converter. This produces a short-term positive pressure that allows emissions to escape the converter mouth.

Fugitive emissions are also a problem during the copper blow, especially when increased air blast is applied at a depth of over 0.5 m in the bath. At that time, the oxidation process in the melt proceeds at a high rate, especially when the bath temperature is 1200° to 1300°C. The rapid rate of reaction and SO_2 evolution is demonstrated by the high oxygen utilization in the blast air (90 to 95 percent) despite the short air residence time in the bath (0.1 to 0.15 s). The evolving SO_2 mixes with the nitrogen in the blast air and the remaining free oxygen, and fills the converter interior near the tuyeres. Fugitive emissions escaping through the mouth result from the pulsating variations in gas volume caused by the dynamic interaction of the blast air and the gases evolving from the melt. The high-velocity gases entrain molten metal droplets, which contain both matte and slag because of the blending of the melt caused by continual blowing. Most of the droplets return to the bath, but others are released through the mouth or carried over into the siphon and ductwork by the pressure pulsations. Table 25 shows the fugitive emissions present at various stages in the operating cycle. Figure 19 shows the position of the converter during charging and pouring operations.

A high blast rate is necessary to achieve adequate throughput, and there is naturally a tendency to increase the blast, which results in a positive pressure in the converter. This practice, however, conflicts with both environmental requirements and the necessity of maintaining a high capture of SO_2 to supply the acid plant. In addition, an upper blast limit cannot be exceeded without causing the agitated melt to splash out of the converter mouth. It is difficult in practice to maintain at all times the slight negative pressure in the converter that would prevent fugitive discharges. Because of the detrimental effects on both the environment and the acid plant, the converter gases cannot be bypassed to an emergency stack either during pressure fluctuations or to prevent excess cooling of the converter during blast reductions.

The only process monitoring at the converter is of blast air rate and pressure, and off-gas temperature and composition. Methods to measure fugitive emissions have not been developed by the design engineers, and fugitive emissions measurements are therefore not made because of the lack of a test method that yields comparable data. This is not a problem unique to Poland. A sulfur balance also does not yield reproducable data. Fugitives are determined indirectly by measuring the levels of metallic elements in the vicinity of the converters. These measurements are conducted regularly; Table 26 presents the results of tests made in 1977. These data show a considerable increase in sulfuric acid concentrations during the slag blow; however, there are no significant variations in SO_2, lead, and zinc because the measurements are averages carried out over a period of time, and therefore cannot reflect instantaneous changes or the continuous variations in fugitives from individual furnaces. More sophisticated measuring equipment located near the converter mouth would be necessary to gain a better understanding of this problem.

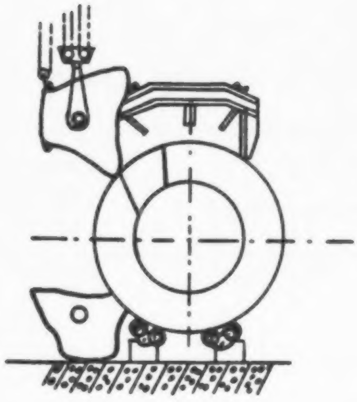
Any estimates of fugitive emissions from the converters must take into account the natural ventilation present in the converter aisle that is upset by emissions from the furnace mouths. The entire air distribution and movement in the area are therefore affected, and any interpretation of monitoring data must

TABLE 25. CHARACTERIZATION OF FUGITIVE EMISSIONS

Converting stage	Fugitive emissions	Total duration of emissions (minutes)
Charging cold additions	None visible	0
Charging first five ladles of matte	Slow fugitive emissions visible shortly after blast is put on to avoid clogging tuyeres; low dust and SO_2 concentration	10-15
Charging sixth ladle of matte and beginning slag blow	Increasing emissions until the gas takeoff system is regulated to the increased blast	2
Slag blowing	None visible	0
Slag pouring	Visible emissions from stream of slag leaving converter; SO_2 and vaporized droplets	10
Copper blowing	None visible	0
Finish blowing[a]	None visible	0
Copper pouring	Very small amount of visible emissions; particulate only	15

[a] Carried to point of 0.5 percent oxygen in matte (5 minutes).

CHARGING



POURING

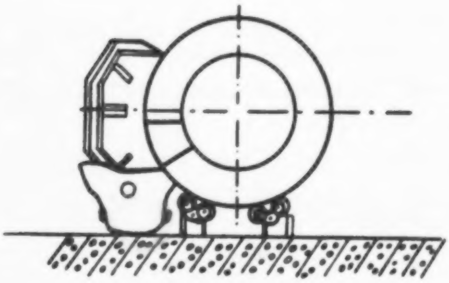


Figure 19. Converter position - charging and pouring operations.

TABLE 26. RESULTS OF AIR SAMPLING NEAR CONVERTERS

	Concentration (mg/m^3)				
	Cu	Pb	Zn	SO_2	H_2SO_4
Standard[a]	0.1	0.05	1.0	20	1.0
Slag blow[b]	0.1343	0.3868	0.3325	37.50	30.00
	0.2690	0.4795	0.3427	70.50	40.00
	0.5243	0.5754	0.3836	87.00	60.00
	0.7097	2.1099	0.5115	-	-
Copper blow[c]	0.0978	0.2250	0.2610	60.0	15.0
	0.1440	0.3720	0.3250	62.0	20.0
	0.4440	0.7190	0.4500	67.5	25.0
	0.7500	0.9380	0.5000	84.0	30.0

[a] Maximum allowable workplace concentration established by Polish Ministry of Health.

[b] Air temperature 12.0°C, relative humidity 74 percent.

[c] Air temperature 15.0°C, relative humidity 63 percent.

consider that instantaneous changes may not be accurately recorded. It is likely that only average values could be obtained, the analysis of which is subject to possible error.

The basic design of the converter assumed an air inleakage of 30 percent of the blast air. This implied that a gas temperature of 950°C in the siphon would result and also an SO_2 concentration of about 8 percent during the slag blow and 11 to 12 percent during the copper blow. In practice, the SO_2 concentrations are lower, probably because of increased dilution air entering through the converter mouth. This increased air inleakage and lower siphon temperature result in the buildup of materials in the siphon, which changes both pressure distribution and gas flow patterns.

The chemical composition of fugitive emissions may be more important than their quantity. The ambient SO_2 level was found to be consistent irrespective of the converter operating cycle, and averaged about 4 times the allowable standard for workplace air. This indicates the need for still further control. The sulfuric acid concentration was far higher than the workplace standard and showed considerably more variation, confirming the presence of fugitive discharges. Both U.S. and Polish smelters share the SO_2-H_2SO_4 problem, but the release of lead compounds is a more serious problem in Poland because of the high lead content of the concentrates. In contrast, there is very little arsenic in Polish concentrates, and therefore no problem with this element.

PRESSURE AND FLOW RATES

Gas pressures and flow rates in the gas cleaning system could be determined by conducting measurements at appropriate points. However, there are problems in conducting such monitoring, including the difficulty of developing instrumentation suited to these operating conditions. Blast air pressure is measured continuously at the tuyeres, and continuous gas pressure measurements are made in the dirty gas collecting chamber, the ESP's, the clean gas collector, the ductwork to the acid plant, before the scrubbers and wet ESP's in the acid plant, and in the single line following the wet ESP's. It is also possible to measure the pressure of the gas when it is at the cooler outlets and after it passes through the acid plant turboblower but instrumentation for continuous measurement of these locations has not been installed. Determination of flow rate is a greater problem, and only the blast air to the tuyeres and the inlet ductwork to the acid plant are now instrumented. The installation of additional continuous flow monitoring equipment is not possible because of the frequent changes of direction in the ductwork and the resulting gas turbulence. A Prandtl tube could be used to determine flow, but this would be a time-consuming and rather inaccurate procedure.

A second problem in determining flow and pressure is the highly variable operating conditions in the system; in particular, the changes which necessarily result from having a different number of converters operating at the same time. In addition, the gas flow through the system varies during each converter's operating cycle, and the leaks in the coolers and ESP's vary between individual converters. The operating conditions in the acid plant change significantly after replacement of the catalyst bed, which in turn causes a substantial change in pressure distribution and flow rates. Emergency situations, operating problems, and equipment failures also affect pressure and flow rates.

Because of these problems, a mathematical model was developed to determine the pressure and flow rates in the converter gas cleaning system. It was developed for two specific cases: operation of a single converter at a blast of 25,000 Nm^3/h; and a two-converter system with blasts of 24,000 Nm^3/h to each furnace. The model was initially developed based upon field test data for pressure and flow at the points where measurements could be made. It was then refined and verified by additional testing. A digital computer of Polish manufacture (ODRA 1325) was employed, using the methodology described in Appendix B, which also presents the results of these model computations.

THE GAS PUMPING CONTROL SYSTEM

The gases from each individual converter pass through the cooler and are forced into the dirty gas mixing chamber by first-stage fans. After the dust is removed in dry ESP's, the gases are collected in a second chamber and are then forced by second-stage fans to the sulfuric acid plant. Operation of the gas pumping system is a complex operation because of the conflicting nature of the discontinuous gas flows from the individual converters and the need for a continuous feed gas to the acid plant. The acid plant is especially sensitive to changes in the inlet SO_2 concentration, which should range over no more than 1 percent by volume. Fluctuations greater than this range reduce the conversion efficiency and result in a tail gas with an SO_2 content that exceeds the 1150 ppm emission limitation for the Glogow No. 1 smelter. A considerable reduction in the feed gas SO_2 content disturbs the heat balance in the contact acid plant, which also lowers conversion rate. Variations in the flow rate of the feed gases cause pressure fluctuations that can possibly lead to gas leakage or dilution in the scrubbing towers.

To provide an acid plant feed gas of steady quantity, pressure, temperature, and SO_2 concentration requires a complex operational and process control system. Many characteristics of the process prevent simple operational and control systems from working satisfactorily. Sudden and large changes in the gas volume from individual converters result from the cyclical nature of such operations as matte and slag pouring, flux addition, and changes in the blast through the tuyeres. Fluctuations also result from clogging and subsequent punching of the tuyeres, and the occurrence of either fugitive emissions or air inleakage through the converter mouth. The situation naturally worsens as the operating cycle proceeds because of the increased hydraulic resistance in the converter. Pressure fluctuations are caused by the interaction of the individual converter gas streams as they mix in the dirty gas collecting chamber. Pressures in the individual ESP's fluctuate because of changes in the gas flow withdrawn to the acid plant, whereas a constant slight negative pressure is desirable to ensure an even gas flow through the ESP and prevent dust accumulation on the insulators. The ESP temperature must be maintained above 270°C to prevent acid precipitation, but below 450°C to prevent thermal damage to the insulators and structural components.

The following conditions are necessary for optimal operation of the gas cleaning system:

Proper scheduling of the operating cycles of the individual converters

Prevention of fugitive emissions through the converter mouth

Air dilution through the converter mouth equal to about 50 percent of the blast air

Gas temperature in the ESP's between 270° and 450°C

Prevention of a positive pressure in the ESP

SO_2 concentration of 6 to 8 percent to the acid plant

Prevention of a positive pressure in the particulate removal circuit in the acid plant

Effective communication between the converter operator and the converter control room, including visual observation of the converter mouth

The Automatic Gas Control System

Automatic process control to simultaneously achieve all these objectives is difficult. The partially automated system operated on the 33-Mg converters proved inadequate for several reasons. Continuous direct measurement of the converter gas volume was not possible because of the high dust loading and entrained slag droplets, high temperature, and the tendency of the lead-bearing dusts to agglomerate and stick to surfaces. Measurements of internal converter pressure was possible, although difficult, as the best location for a sensor was near the furnace mouth, where it could be easily damaged during charging operations. Converter off-gases could not be controlled by valves in the ductwork because they soon became clogged with the sticky dusts. The system pressure could not be adequately balanced because changes in the gas flow from one converter caused pressure changes in the dirty gas collector that were transmitted to the other operating converters. A major factor which contributed to the difficulties of automatic feedback control was related to the size and configuration of the system, in which large pieces of equipment are connected in series or parallel by extensive ductwork. It proved slow to respond to changes in gas volume because of the length of the duct runs.

Automatic controls for the gas cleaning system on the 80-Mg converters have been designed based upon the experience with the partially automated system on the smaller converters, and many of the problems listed above have been solved. A more sophisticated control system was desired, and it was assumed that improvements could be made that would reduce the disturbances and upset conditions that routinely occur. Figure 20 is a diagram of the system, using Polish instrumentation designations. Automatic controls would be provided at the following points:

- Blast air to the converter
- Cooling air intake
- Gas pressure in the dirty gas collector
- First-stage fan
- Second-stage fan
- Gas pressure in the acid plant wet ESP's
- Gas flow in the acid plant

Supplementary automatic override controls would also be provided to rotate the converter from its operating position in event of loss of blast, and also controls to monitor and synchronize the individual elements in the system, punch tuyeres, and correct the gas temperature at the cooler outlet. All controls would be driven by electrically-signaled compressed air servomotors, which have proved reliable in spite of the difficult operating conditions. The following paragraphs describe the operation of each

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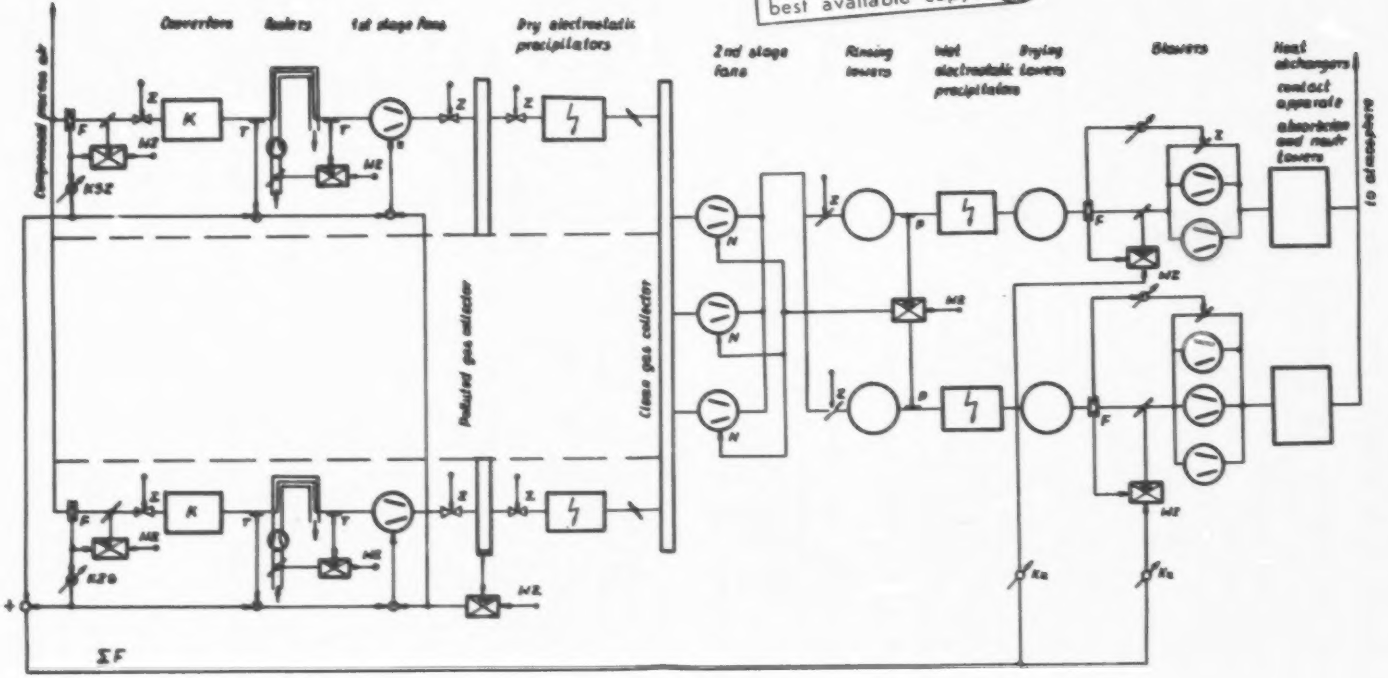
Compressed process air
Converters
Boilers
1st stage fans
Dry electrostatic precipitators
Polluted gas collector
Clean gas collector
2nd stage fans
Rinsing towers
Wet electrostatic precipitators
Drying towers
Blowers
Heat exchangers
contact apparatus
absorption and mist towers
to atmosphere
K
F
T
Z
Nz
KSZ
KZO
N
p
Ku
ΣF
+

__Designations:__ F - flowrate ; ku - coefficient of share ; kss - suction coefficient ; N - rotations ; p - pressure ;
T - temperature ; Nz - desired value ; Z - position
● - adding element ; ● - multiplying element ; ϕ - adjustable multiplier ; ⊠ - proportional controller ;
•—• remote control.

Figure 20. Proposed automatic gas control system.

element in the automated control system, which has been only partially implemented at the Glogow No. 1 smelter.

The converter blast intensity control must both stabilize the blast and reduce disturbances that arise during tuyere punching. A conventional proportional controller of constant proportional gain is used because the integrating elements of PI (proportional-integral) or PID (proportional-integralderivative) cannot respond quickly to frequent converter rotations. The proportional controller can react immediately, and the system should work well, providing automatic tuyere punching is provided to prevent unstable blast flows because of tuyere clogging. The flow rate is monitored with an orifice plate and transmitter on the compressed air line to each converter. The flow controller also receives an override signal from fan speed measurement. The final control valve with its servomotor is placed on the compressed air line at the converter in front of an emergency shutoff valve. Both control valves can be activated either manually or automatically from the control room.

The amount of cooling air pumped through the forced-air coolers is adjusted by a damper at the fan inlet, which is also driven by a pneumatic servomotor. The damper is actuated by a temperature controller in the control room, which receives its input signal from a sensor at the cooler outlet.

Interacting systems are used to control the first-stage fan speed and the gas pressure in the dirty gas collector, thus assuring both proper flow from each converter and a constant pressure in the collector. The fan speed is related to blast intensity, and the speed of each fan is controlled in ratio to the measurement of blast air flow to the corresponding converter. This is a valid relationship, since all system pressures are very close to atmospheric, and over the range of normal operation, fan speed and flow rate are directly related. These signals are further modified, however, by the output of a PID controller whose input is from a transmitter measuring pressure in the dirty gas collector. Therefore, all fans are simultaneously increased or decreased in speed to maintain constant pressure downstream of the gas coolers. After trimming, these signals are used to generate a signal which is a principal input to control the speed of the acid plant turboblowers, as described in a following paragraph.

The signal to each individual fan is trimmed once again using a manual multiplying device. These final trimmers compensate for variations in resistance in the ductwork from individual converters, to maintain equal degrees of dilution air through each of the operating converters. It is sufficient to make these adjustments manually once every several days because the resistance changes slowly. To gauge the desired degree of compensation, the converter mouth must be visually monitored; the best approach would be a control room with a clear view of the converter aisle or closed-circuit television. This is the most notable omission now in the Glogow No. 1 smelter.

Final control of fan speed is through the SCR (silicon controlled rectifier) drive of the direct current fan motors, with feedback from electronic speed measurements.

The temperature of the off-gases from the individual converters, which varies with the operating cycle, is also adjusted by signal to the first-stage fan regulators. This compensation is made automatically in response to temperature changes measured at the cooler inlet. To eliminate the momentary delay that would occur in the temperature sensor when the converter blast is first made, an overriding minimum temperature device is built into the system that shuts off automatically when the sensor reaches that level.

The acid plant wet ESP's are stabilized automatically at atmospheric pressure by adjustments to the second-stage fans. The PI controller on these fans is actuated by a signal from a pressure sensor in the collecting chamber located in front of the ESP's. Each fan's input signal is multiplied by the same adjusting coefficient; these fans and their drive mechanisms are identical to the first-stage fans. The pressure controller is located in the control room.

The gas sent through the acid plant must be diluted with air to maintain SO_2 concentraion between 6 and 8 percent. The total flow of acid plant gas is also related to the total off-gas flow from the converters. As previously described, signals are available which represent flow rate from each converter. The necessary degree of dilution varies, however, depending on the blowing stage an individual converter is in. During slag blow, dilution ratio is 2 to 1, but during the copper blow, when more SO_2 is generated per unit volume of air, the ratio is 2.5 to 1. Instrumented signals are therefore generated using multiplying relays in the control room which are manually adjusted to conform to each converter's blowing stage. These signals are then totalized with a summing device, and the output, with suitable scaling, represents the desired total flow rate of offgas plus dilution air to be sent to the acid plant. This signal becomes the set point to a PI controller which adjusts a pneumatically-actuated damper in the turboblower suction. Flow of total gas to the acid plant is measured with a venturi tube in a vertical section of ductwork upstream of the turboblowers, thereby providing positive feedback.

Surge of the turboblowers is prevented by a recycle line containing a control valve which bypasses from discharge to suction to maintain a pre-set minimum flow rate. The bypass valve is actuated when needed by a proportional-only controller receiving an input signal from the venturi transmitter.

The converter must be automatically rotated to prevent tuyere plugging in the event of insufficient blast. The converters now installed rotate when the pressure in the tuyeres drops below 0.4 kg/cm^2. It has been observed, however, that after punching, the pressure does not rise because the tuyeres are out of the bath and there is no resistance to the blast. As a result, the converter remains in the rolled-forward position until the automatic system is overridden manually. To prevent this problem, blast flow rate is to be added as a control parameter to the system. After cleaning, when blast flow is high, the converter will be automatically returned to the vertical position in spite of the low pressure caused by the lack of resistance. Measurement of blast flow rate alone cannot provide adequate control, however, because the plugging of a single tuyere could then result in a perceived loss of blast that would cause automatic rotation. Control by both pressure and flow is thus required.

An automatic tuyere puncher of Polish design (patent PRL 90615; May 26, 1976) has been installed on the tuyeres to eliminate the necessity for manual punching and to allow more stable conditions in different operating stages. The puncher is activated by a signal when a specified level of pressure is reached in the tuyeres. The puncher operation is flexible, and it can be programmed to clean only every second, third, or fourth tuyere if desired.

The interactive system of controls described in this section allows synchronized operation of the converters, bypass stack, parallel ductwork lines, first-stage fans, and gas flow. The overall system would reduce disturbances or upsets, fugitive emissions, and excess air inleakage, as well as simplify monitoring and control by the smelter personnel. Of particular value would be the system's ability to react to rapid changes in blast intensity without the delays that are characteristic with the present manual control of the fans in a system of this size.

As this section has indicated, the design and operation of a converter gas cleaning system are complex. Even a sophisticated control system is unable to overcome the limitations of a poorly designed or operated installation. Proper synchronization of the converter operating cycles is essential. It would not be possible to have two of three converters simultaneously rotated forward for an extended period of time. The best approach to the design of an automatic control system must be based on feed-forward of gas flow through the acid plant from the sum of the blast air flows to the individual converters. Such a system eliminates the necessity for any monitoring except temperature in the converters or associated ductwork. It does not require the use of frequently unreliable gas analyzers to control air dilution or SO_2 concentration. Proper operation of the ESP's is assured by the controls in the dirty gas mixing chambers and the wet ESP inlets in the acid plant. The system operates with continuously variable fans, and does not rely on the use of valves in the ductwork. The only valves now in use control the blast air and the turboblowers, and they are operated by reliable servomotors driven by compressed air. The off-gases from individual converters are controlled by use of the unique first-stage fans, whose speed is modified by various compensating coefficients. This shifts the responsibility for their proper operation from a single operator to an integrated approach that includes the whole system.

CONVERTER PRESSURE SENSOR

Pressure fluctuations in the converter cause either fugitive emissions or air inleakage through its mouth. Because the mouth has a diameter of only 1.5 m, changes in pressure of only several millimeters of water can have a considerable effect. There is, of course, no possibility of closing or blocking the converter mouth because of the necessity of frequently charging matte, flux, and cold scrap, as well as taking blister samples for analysis. The pressure at the mouth should therefore be kept as low as possible without creating excess air inleakage.

Pressure measurements within the converter are also difficult to make, however. Temperatures are high and fluctuating, and the furnace atmosphere is unstable and reactive. Considerable fume and entrained particulate matter are in the gases. Variations in the blast through the tuyeres and the amount of obstruction in the siphon and gas removal system further affect the internal pressure distribution. In addition, the pressure at the mouth changes during converter rotation and charging and pouring operations, during which time there is also the possibility of the molten bath splashing onto an instrument. A pressure sensor mounted inside the converter must therefore be able to handle very difficult conditions, and still provide accurate measurements over a converter campaign. For accurate pressure measurements, a sensor must be located as near as possible behind the converter mouth, as shown in Figure 21. At this location, it is protected from direct contact with the molten bath or from splashing even during full forward rotation or charging. A steel sheet around the mouth can provide protection from damage by the cold scrap additions.

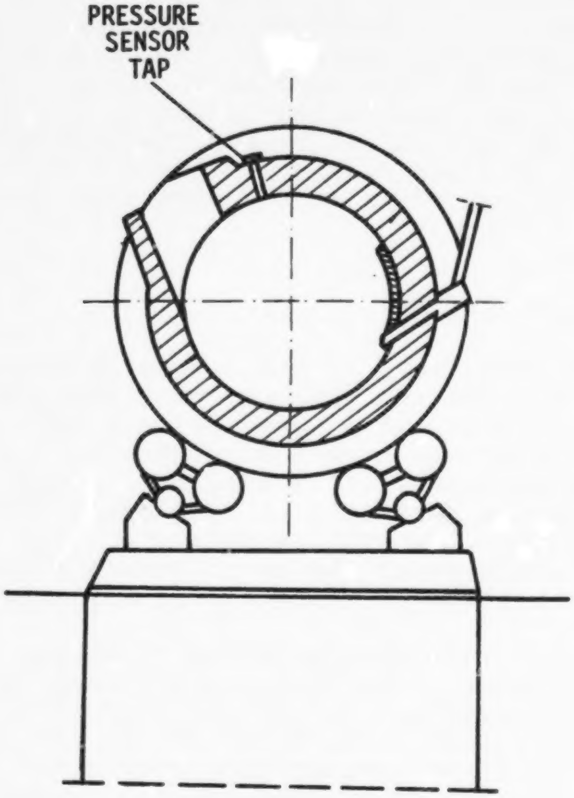
PRESSURE
SENSOR
TAP

Figure 21. Location of converter pressure sensor.

During 1978, a pressure sensor was operated for 6 months on one converter. The probe was successful in monitoring the internal pressure near the converter mouth. Both these pressure measurements and the blast air rate were recorded by continuous monitors; representative strip charts are shown in Figure 22. The probe did not require additional maintenance, and it functioned properly throughout an entire campaign. It was not obstructed by deposits, but instead the refractory lining gradually wore away. During normal converter operations, the maximum pressure fluctuations were only a few millimeters of water. To approach the goal of zero exchange of ambient air and furnace gases would require that these fluctuations be less than 1 mm H_2O. The test program demonstrated that changes in the blast rate have a considerable influence on the pressure at the mouth; an increased blast requires a corresponding increase in negative pressure to prevent fugitive emissions. It was also determined during the test program that the first-stage fans could be used to stabilize the converter pressure.

During the probe test, the amount of negative pressure required to minimize fugitive emissions was determined by visual observation of the converter mouth. The measured value can also be confirmed by calculations. Assuming a blast ranging from 15,000 to 30,000 Nm^3/h and a bath surface of about 20 m^2 (7.6 m x 2.8 m), the velocity of the gases leaving the bath is as follows:

$$V = \frac{Qd}{A} = \frac{15{,}000 \text{ to } 30{,}000}{20} = 760 \text{ to } 1500 \text{ m/h} = 0.208 \text{ to } 0.416 \text{ m/s}$$

Assuming a gas density of 1.5 kg/m^3, the pressure can be found using Bernoulli's equation:

$$P_d = \rho \frac{V^2}{2} = \frac{1.5 \text{ x } (760 \text{ to } 1500)^2}{2} = 0.32 \text{ to } 1.3 \text{ mm } H_2O$$

The assumptions are conservative, and the negative pressure should be kept at about 1 mm H_2O. The pressure measured with an accurate sensor near the converter mouth could be compared with atmospheric pressure near the converter, with the difference used as a control factor for either the blast or the first-stage fan. This represents an alternate control approach to the feedforward system described previously.

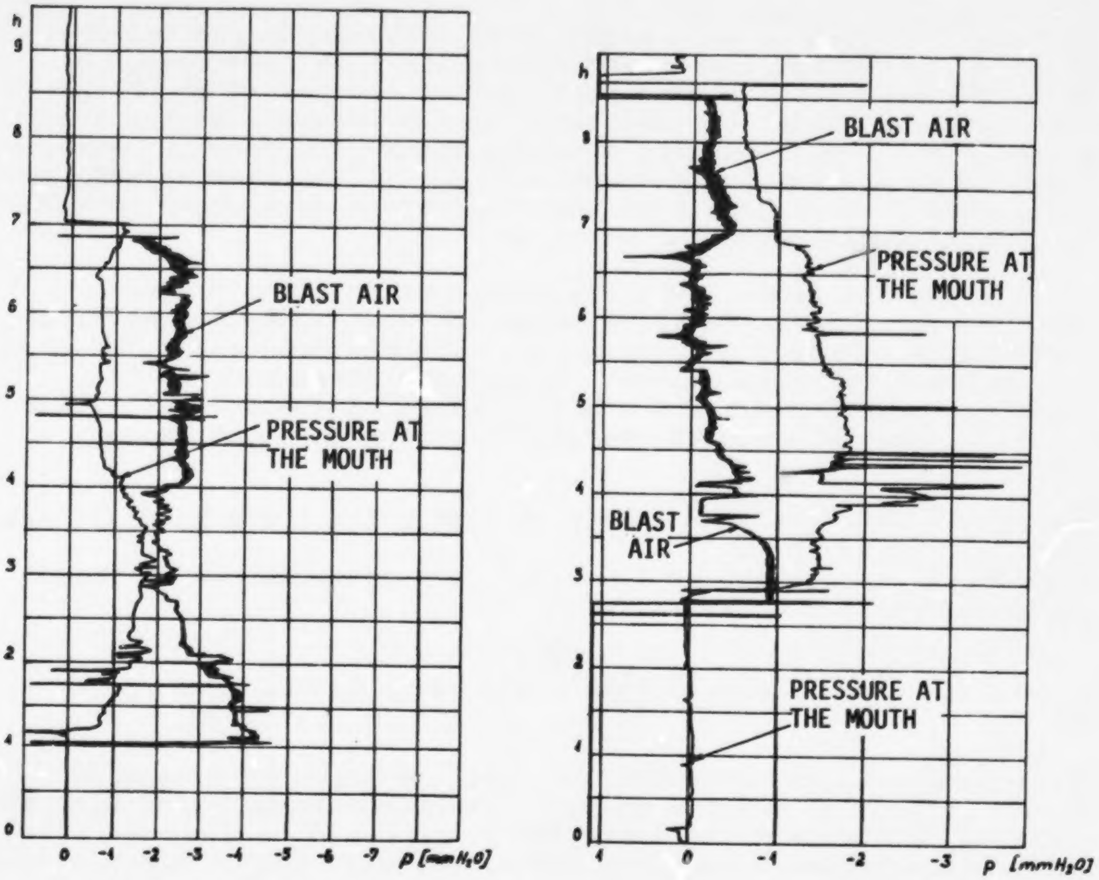

Figure 22. Strip charts - converter blast and pressure at mouth.

SECTION 4

APPLICATION TO U.S. PRACTICE

As part of the Polish responsibility associated with this research project, it is necessary to discuss how the procedures described in the previous sections could be applied to the various types of copper smelters encountered in United States industry. Although many of the techniques used in Poland differ from conventional U.S. practice, most should be directly transferable.

The Polish copper industry has had considerable success with the design, construction, and operation of the siphon converter. Industrial units with capacities of 33 and 80 Mg of copper per operating cycle have been employed for several years. The converters have been equipped with several unique features of Polish design, including automatic tuyere punchers, flux feeders with automatic weighers, and a partially automated gas control system.

When compared with the off-gases from Peirce-Smith converters processing U.S. concentrates, the off-gases from the Polish siphon converters have considerably higher SO_2, SO_3, and lead contents. The quantity of arsenic at the Polish converters is much lower. The higher SO_2 content is a result of the reduced air filtration into the system. The acid dew point is very high (270°C) because of this high SO_2 concentration and the SO_3 formed during gas cooling. A comparison of the off-gas characteristics of the two converter designs is presented below:

	Peirce-Smith	Siphon
Dilution air	150 percent	50 percent
Gas temperature	600° to 800°C	800° to 900°C
SO_2 concentration	6 to 8 percent	10 to 14 percent

Despite differences in present operation as compared with U.S. practice, no problems are expected in adapting the Polish converter design to installations that process other types of concentrates. Although matte containing 55 to 67 percent copper and 8 to 14 percent iron is now processed, in the past matte with a lower copper content (50 to 55 percent) and higher iron content have been successfully converted. White mattes have also been successfully processed in the siphon converter. Oxygen enrichment of blast air is necessary only when the charge is composed of a high-copper low-iron matte and considerable cold scrap (50 to 60 percent of matte quantity).

A comparison of the operating procedures cannot be made at this time because of a lack of experience in Poland with the Peirce-Smith converter. However, a stated U.S. objection to the siphon converter is that the relatively small diameter of the converter mouth causes problems with charging matte, cold dope, and anode slag, or skimming slag and pouring copper. Because spent anodes at Glogow No. 1 are sent to a refining furnace, a converter with a large diameter mouth is not needed. A larger mouth would naturally permit the charging of larger pieces of cold dope. It is not believed that an increase in size would influence the production capacity or proportion of cold additions required for efficient operation; however, it would increase fugitive emissions. The lengthwise diameter of the mouth is most critical in this respect. The pressure in the converter varies from negative at the siphon, to approximately zero at the mouth, and positive at the far end wall, as shown in Figure 23. Increasing the mouth size toward the end walls upsets this balance by exposing a larger portion of the furnace interior that is operating under positive or negative pressure. To prevent such an imbalance from occurring, an elliptical design should be installed if a larger mouth size is desired.

After many years of experimentation, a particulate and SO_2 control system has been developed that consists of gas coolers, dry ESP's, and an acid plant with wet gas cleaning, all synchronized by a unique two-stage gas pumping system. This system provides a gas stream of stable volume, temperature, and composition to the ESP's, thus assuring their proper operation. It allows continuous gas withdrawal from the converters, regardless of the number under blow, and is flexible enough to allow individual ESP's to be bypassed if repairs are needed. The capital costs are also lower than would be the case if each converter had a separate particulate control system. Sampling and analytic work has demonstrated that the use of this control system is largely responsible for the stable gas characteristics.

Although the gas coolers still suffer some operational damage, the newest design has proved to be more efficient, stable, and flexible than those previously employed. This cooler could be readily adapted to the Peirce-Smith converter with some modifications. Because of the lower gas temperature and volume, there is less chance for damage to the cooler with a Peirce-Smith converter. In adapting the cooler design, there would be no need for the initial atmospheric cooling section that is designed to protect the remaining forced-air sections from damage. The size of cooler necessary for this application must be defined for

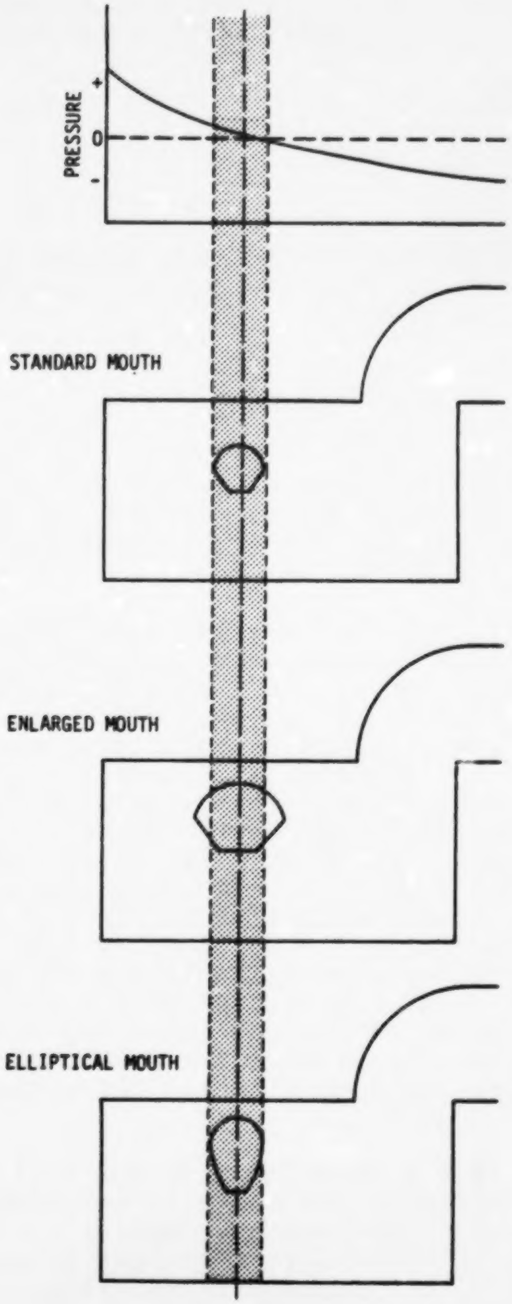
PRESSURE
+
0
-
STANDARD MOUTH
ENLARGED MOUTH
ELLIPTICAL MOUTH

Figure 23. Effects of increased converter mouth size.

the gas parameters which influence the rate of heat exchange. In comparison to U.S. practice, the effects of Reynolds numbers and mean logarithmic temperature difference cancel each other, as follows:

$$\frac{R_1}{R_2} \times \frac{\Delta T_{m1}}{\Delta T_{m2}} \cong 1$$

The amount of cooler surface needed thus depends primarily on the amount of heat to be taken from the gas to reach the desired temperature (400°C), as follows:

$$\frac{S_2}{S_1} = \frac{600-400}{800-400} \times \frac{R_1}{R_2} \times \frac{\Delta T_{m1}}{\Delta T_{m2}} \cong 0.5$$

The cooling surface required for a Peirce-Smith should therefore be about half that necessary for a comparable siphon unit. The flow cross-section of the gas cleaning system must be increased, however, because of larger gas volumes.

To prevent excess internal gas velocity, there should be 1.5 to 2.0 times as much ESP capacity with the Peirce-Smiths. The average particulate loading at the Glogow No. 1 ESP inlets is 8 g/Nm^3. An outlet loading of at least 0.25 g/Nm^3 can be obtained if the ESP's are operated at 50 kV with a corona discharge of 300 to 370 mA for two fields. The average loading behind the ESP's is 0.21 g/Nm^3, which represents a cleaning efficiency of 95 to 99 percent (average 97.24 percent). Since all gas is scrubbed at the acid plant, better efficiency is unnecessary. The gas temperature in the ESP's must be held above the acid dew point to prevent acid condensation that will coat the electrodes with dust and lead to reentrainment of the particulate and a loss in cleaning efficiency. ESP's are also suitable for use with Peirce-Smith converters, although 1.5 to 2.0 times greater capacity would be required because of the increased gas volume. Because of the dangers of acid precipitation, a preheater should be used to prevent the temperature from dropping below the acid dew point during startup.

The efficiency of SO_2 absorption in the dual-contact acid plant is about 99.0 percent, and the acid emissions in the tail gas are within the prescribed limits. Sulfur losses in the acid plant are thought to be about 3 to 6 percent of the total sulfur loss, including from 1 to 2 percent in the blowdown slurry. Increased sulfur control is possible, but would require more efficient operation of the gas cleaning system. This would entail an even more carefully planned converter operating schedule, use of more modern ESP's, and implementation of a fully automatic gas control system based upon the principles outlined in this report. With appropriate modernization and design modifications, such a system could also be installed on a U.S. copper smelter using either siphon or Peirce-Smith converters.

Proper performance of this system requires a program of maintenance to minimize leaks of air into the system. Prevention of leaks also requires operation of the gas removal system within the proper temperature range, attention to gas shut-off devices such as valves and flaps, and in particular, maintenance of airtight seals between the dust hoppers and the coolers, collecting chambers, and ESP's. Leaks in the dust removal system from the ESP's seem to have the greatest effect on the dilution problem (where negative pressure up to 20 mm H_2O can develop). In addition, leaks at this point reduce ESP efficiency as the settled dusts become reentrained in the gas. Leaks in the gas cooling and cleaning system can also result from improper choice of materials of construction, as well as poor design that fails to provide for the thermal stresses that will be encountered.

SECTION 5

RECOMMENDATIONS FOR FURTHER RESEARCH

Several unresolved questions still remain concerning the design of the siphon converter and the operation of the gas cleaning system. The following work areas concern the converter itself:

- The physical and chemical characteristics of the aerosol produced when an oxygen-enriched blast is employed (especially at concentrations greater than 27 percent oxygen), and the effects on refractory life
- The design of the tuyeres, their angle of inclination into, and effect on, the bath, and the degree of oxygen utilization in the converting process
- The effect of flux composition on the removal of trace metals from the matte and the composition of the furnace off-gases.

The following investigations could be made with respect to the optimization of the gas cleaning system and the sulfur recovery process:

- SO_3 formation in the converter, and techniques to prevent or limit this process
- Fugitive emission sampling and analysis
- Trace element emissions to air, water, and solid waste
- Implementation of the automatic control system desribed in this report and assessment of its effect on sulfur recovery

BIBLIOGRAPHY

Bystroń and Małusecki. "Świeżenie bogatego kamienia miedziowego." (Refining of Copper Matte.) Rudy i Metale Niezelazne. Nr. 6, 1965 r.

Bystroń and Pieczka. "Zagadnienie świeżenia bogatych kamieni miedziowych w konwertorach typu Hoboken." (Problem of Refining Rich Copper Matte in Hoboken Converters.) Sympozjum Nauk-Techn. Nowoczesna Metalurgia Miedzi. IMN, Komitet Hutn. PAN, Komb. Górn.-Hut. Miedzi t.I.

Czernecki, Śmieszek, and Sobierajski. "Usuwanie Olowiu z miedzi w procesach świeżenia." (Lead Removal from Copper During Refining.) Prace IMN. 1975, t.IV.

Dalewski, Ozóg, and Skrzyś. "Opyt ekspluatacii suchich elektrofiltrow w cvetnoj metallurgi PNR." (Experience in Operating Dry ESP's in Nonferrous Metallurgy in Poland.) Bjull. SEV: Nauc.-Tech. Soveta po Ochr. Prir. 1977 r., nr. 3.

Grabowski and Kowal. "Konwertorowanie bogatych kamieni miedziowych." (Converting Rich Copper Matte.) Sesja Naukowo IMN "Rozwój nowoczesnych technologii w przemyśle metali niezelaznych." Gliwice, 1977. t. III, cz. 1.

Mścichowski, Peszat, and Babik. "Zagadnienie odpylania w hutnictwie miedzi gazów konwertorowych, kierowanych do fabryk kwasu siarkowego." (Problems in Particulate Removal in Gases Supplied to a Sulfuric Acid Plant.) Miedzynarodowe Sympozjum Nauk-Techn. Nowoczesne Procesy w przemyśle met. niezelaznych. OPT Katowice, 1978.

Ozóg and Peszat. "Skuteczność elektrofiltrów odpylajacych gazy z konwertorowania kamienia miedziowego." (Efficiency of ESP's in Particulate Removal from Converter Gases.) Rudy Metale 1977 t.22, nr. 2.

Sobierajski and Czernecki. "Świezenie smopu miedz-ołów-zelazo w konwertorze." (Refining of Copper-Lead-Iron in a Converter). Sesja Naukowo IMN "Rozwój nowoczesnych technologii w przemyśle metali niezelaznych." Gliwice, 1977. t. III, cz. 1.

Sobierajski and Śmieszek. "Zagadnienie ogniowej rafinacji miedzi od zanieczyszczeń Metalicznych." (Problems of Metallic Impurities in Copper Refining.) Sesja Naukowo IMN "Rozwój nowoczesnych technologii w przemyśle metali niezelaznych." Gliwice, 1977. t. III, cz. 1.

Weglorz and Warczok. "Rozwój techniki konwertorowania kamienia miedziowego." (Development of the Technique of Converting Copper Matte.) Bjull. IMN 1970, nr. 3.

Wiechecki and Kopeć. "Automatyzacja i mechanizacja wybranych procesów technologicznych." (Automation and Mechanization of Some Manufacturing Processes.) Sesja naukowo IMN "Rozwój nowoczesnych technologii w przemyśle metali niezelaznych." Gliwice, 1977. t. III, cz. 1.

APPENDIX A

OTHER CONVERTER STUDIES

LIME ADDITION TO CONVERTER GASES

The off-gases from the siphon converters contain from 5 to 13 percent SO_2 (volume) at the ESP inlet. In addition, some SO_3 is formed within the converter and ductwork. Table A-1 presents the results of SO_2-SO_3 testing of the converter gases, and it indicates that the ratio of SO_3 to SO_2 ranges from 0.018 to 0.078 (average 0.035). The SO_3 content of the gases is of particular importance because it is the principal determinant, along with the steam content of the gases, of the acid dew point. Drops in the gas temperature below this point (270°C) are undesirable because of the resulting considerable increases in corrosion within the system. A test program was conducted to determine the effects of adding CaO or $Ca(OH)_2$ to the converter gases to reduce this problem.

Preliminary laboratory tests were first conducted to investigate the possibility of improving the physical properties of the converter dusts. When $Ca(OH)_2$ was added to a simulated converter gas stream in quantities equal to 20 percent of the total dust loading, the angle of repose was found to decrease from 48 to 40 degrees. This indicated a reduced level of acid precipitation.

Full-scale tests were then carried out at a 33-Mg converter by charging lime by gravity feed through the emergency stack into the ductwork between the converter and the gas cooler. The lime quantity was 10 percent of the converter dust loading. The test ran 44 hours, and was halted when it had become evident that the dusts were not sliding down into the hoppers. Comparative analyses of the dusts taken from the hoppers are presented in Table A-2. It was not possible to draw samples from the first cooler hopper because of sintering of the dusts and lining, but the pattern evident from the second cooler and ESP hoppers is clear. The increase in CaO content in these dusts is very small in relation to the amount added. The majority of the lime simply settled out of the gas stream in the first cooler hopper. The free sulfuric acid content in the samples from the ESP hoppers was then determined, with results as follows:

TABLE A-1. SO_2/SO_3 CONTENT OF CONVERTER GASES AT ESP INLET

	Converter blast (Nm^3/h)	Temp. ESP inlet (°C)	SO_2 content		SO_3 content		SO_3/SO_2 ratio
			g/Nm^3	%	g/Nm^3	%	
1	20,000	420	389	12.10	9.04	0.22	0.018
2	22,000	400	191	5.90	17.40	0.42	0.071
3	28,000	285	251	7.72	6.41	0.15	0.019
4	26,000	325	307	9.47	17.84	0.44	0.046
5	27,000	375	421	13.01	13.39	0.33	0.025
6	26,000	380	378	11.67	15.17	0.37	0.032
7	18,000	385	399	12.31	16.34	0.49	0.032
8	20,000	325	257	7.93	12.95	0.36	0.045
9	22,000	325	296	9.12	10.27	0.28	0.031
10	22,000	325	286	8.80	12.38	0.34	0.038
11	24,000	325	307	9.47	14.62	0.41	0.043
12	24,000	325	307	9.47	17.10	0.48	0.028
13	21,000	360	362	11.15	23.11	0.68	0.050
14	27,000	400	236	7.27	20.54	0.57	0.078
15	32,000	400	278	8.59	18.44	0.51	0.059
16	30,000	400	318	9.80	11.13	0.31	0.032
17	26,000	320	197	6.37	4.13	0.11	0.017
18	25,000	350	228	7.55	7.10	0.20	0.026
19	25,000	380	322	10.44	9.88	0.27	0.026
20	22,000	400	405	12.80	11.40	0.31	0.024
Avg.	24,000	360	320	9.55	13.43	0.36	0.035

TABLE A-2. CaO CONTENT IN HOPPER DUSTS

Sample location	CaO content, %	
	Before test run	After test run
Cooler, second hopper	0.37	0.92
ESP, first field hopper	0.10	0.96
ESP, second field hopper	0.13	0.57
ESP, third field hopper	0.11	0.26
ESP, fourth field hopper	0.20	0.34

- First field 0.93 percent
- Second field 2.86 percent
- Third field 2.94 percent
- Fourth field 4.14 percent

Stoichiometric calculations indicate that only about 3 percent of the total lime added is required to bind the 4.14 percent sulfuric acid found in the fourth field.

The next area of investigation was to determine if lime addition could be used to tie up the SO_3 in the gas stream and lower its acid dew point. Assuming a CaO content of 60 percent for calcined lime and 67 percent for hydrated lime, the quantities required to completely neutralize the SO_3 were calculated according to the following equations:

$$CaO + SO_3 \rightarrow CaSO_4$$

$$Ca(OH)_2 + SO_3 \rightarrow CaSO_4 + H_2O$$

The results are presented in Table A-3.

In the actual field tests, lime was supplied from an 18-Mg tank and sprayed with compressed air into the ductwork between the dirty-gas mixing chamber and the ESP on an 80-Mg converter. The gas temperature in the ESP was held below 270°C by increased use of cooling air. The acid dew point could not be measured during the test because of equipment problems, and as a result, the effectiveness of the lime addition could only be evaluated by the formation of sulfuric acid mist on a steel rod. In addition, the lime feeding apparatus was prone to damage. Thus the observed results can only be considered preliminary. Neutralization was not observed when lime was added at a rate of 40 kg/h with a gas flow through the ESP of 20,000 Nm^3/h. When the lime addition was increased to 160 kg/h (with gas temperatures of 230°C at the ESP inlet and 180°C at the outlet) there was no acid mist observed in the gas discharged from the ESP. With a gas flow of 20,000 Nm^3/h and a particulate loading of 10 g/Nm^3, this quantity of lime is equivalent to 80 percent of the total dust in the gases. Total daily lime requirements at this rate would be 3.84 Mg. As indicated in Table A-4, monitoring conducted during the test program showed no increase in the ESP outlet particulate loading as a result of the lime addition. Samples were also drawn from the ESP hoppers and analyzed for lime and lead content; the results are presented in Table A-5. The coarser lime particles deposited primarily in the first hoppers, which caused some dilution of the lead content of these dusts.

TABLE A-3. STOICHIOMETRIC LIME REQUIREMENTS FOR SO_3 NEUTRALIZATION

SO_2 content		Calcined lime requirement (g/Nm^3)	Hydrated lime requirement (g/Nm^3)
g/Nm^3	%		
2.93	0.08	3.42	4.05
4.27	0.10	4.98	5.90
7.10	0.20	8.19	9.80
10.68	0.30	12.45	14.70
14.41	0.40	16.83	19.90
18.37	0.51	21.41	25.30
21.90	0.61	25.60	30.30
28.68	0.71	33.60	39.80
39.03	0.96	45.50	53.90
44.28	1.10	51.60	61.00
47.83	4.17	55.80	66.00

TABLE A-4. ESP EFFICIENCY DURING SO_3 NEUTRALIZATION TESTING

	Lime feed (kg/h)	Gas flow (Nm^3/h)	Particulate loading (g/Nm^3)		Cleaning efficiency %	Gas temperature (°C)	
			Inlet	Outlet		Inlet	Outlet
1	240	14,900	18.3	0.31	98.30	281	260
2	240	13,100	16.5	0.31	98.12	300	275
3	274	15,100	17.8	0.36	97.98	277	244
4	220	13,700	17.1	0.48	97.19	272	245
5	189	14,800	16.2	0.36	97.78	278	263
6	300	18,900	18.9	0.29	98.47	272	267
7	300	19,700	19.9	0.31	98.44	275	268
8	270	21,800	18.3	0.37	97.98	290	259
9	270	17,100	17.8	0.27	98.48	279	271
10	240	24,200	13.5	0.36	97.33	264	262
11	200	24,200	14.3	0.21	98.29	293	283
12	171	24,300	17.8	0.24	98.65	370	315
Avg.	242	22,200	17.2	0.32	98.08	287	267
13	None	24,200	11.8	0.24	97.96	385	314
14	None	26,100	10.52	0.52	95.05	300	290
15	None	18,800	8.92	0.18	97.98	355	303
Avg.	-	23,033	10.41	0.31	96.99	346	302

TABLE A-5. LIME AND LEAD CONTENT IN DUSTS FROM ESP HOPPERS

Sampling location	Test 1		Test 2		Test 3		Test 4	
	CaO (%)	Pb (%)	CaO (%)	Pb (%)	CaO (%)	Pb (%)	CaO (%)	Pb (%)
First hopper	9.32	46.2	13.12	38.1	16.59	38.3	27.46	27.5
Second hopper	6.24	48.9	10.60	48.8	6.72	48.8	8.62	36.7
Third hopper	2.22	52.3	3.15	49.7	1.19	47.5	3.25	38.9
Fourth hopper	0.23	51.6	0.76	48.3	0.02	46.2	1.00	35.1

Because of the high lime requirements and the problems with the charging system, further tests at the smelter were cancelled. It is believed that lime addition is a feasible method of preventing sulfuric acid condensation, but that maintenance of gas temperature above the acid dew point is a more economical alternative. Lime addition can only be recommended in emergency situations such as a sudden reduction in gas flow.

OXYGEN ENRICHMENT

The most important factor in increasing the efficiency of the converting process is the amount of oxygen available in the furnace. Some plants enrich their blast air with oxygen, which requires the addition of large amounts of solid copper scrap or concentrate to prevent excess temperatures and to protect the refractory lining. Tests were conducted with up to 27 percent enrichment of the blast air, and it was found that solid copper scrap additions of up to 60 percent of the quantity of matte were needed. Oxygen enrichment also increases the SO_2 content and reduces the volume of the off-gases, which greatly facilitates gas removal, cleaning, and sulfur recovery. The effects of oxygen enrichment can be calculated. If a constant quantity of dilution air is assumed to enter through the converter mouth, the SO_2 content of the off-gases will increase in direct proportion to the additional oxygen content of the blast air. This effect is illustrated by the data in Table A-6. A constant quantity of blast air decreases the time needed for converting a given amount of matte in direct proportion to the degree of oxygen enrichment. The total amount of oxygen in the blast cannot change if the converting time remains the same; a proportional decrease in blast air is required, as the table shows.

TABLE A-6. CALCULATED EFFECTS OF OXYGEN ENRICHMENT

O_2 concentration-blast air (%)	SO_2 concentration-off-gases (%)	Blast air requirements (Nm^3/hr)	Off-gases (Nm^3/hr)
20.9	15.0	24,000	29,200
23	15.9	21,900	26,600
27	17.7	18,700	22,700

APPENDIX B

PRESSURE AND GAS FLOW CALCULATIONS

The converter and its associated gas cleaning system are capable of being studied using modeling techniques. Among the processes that occur are gas pumping and storage, heat exchange, chemical reactions, and particulate removal. The gas handling system must operate in a manner that provides the sulfuric acid plant with a feed gas of reasonably consistent volume, pressure, temperature, SO_2 content, and particulate loading. Since changes in gas flow at one point in the system affect all other points, the whole sequence from converter to the heat exchangers and gas preheater in the acid plant must be considered in modeling. Ideally, all of the gas parameters should be part of a model; however, this is an extremely complex undertaking.

The development of a model of the Glogow No. 1 smelter was limited to the gas pumping and storage system because of its importance to the interactions among the system controls. The goal in developing the model was to define changes in pressure and gas flows through the converter and gas cleaning apparatus as the converter operated through its cycle. Pressures in the system are the principal determinant of the air dilution of the gases and fugitive emissions, which in turn influence the gas volume and SO_2 content to the acid plant. Heat exchange phenomena, flow resistance changes caused by temperature fluctuations, chemical reactions, and particulate removal were excluded from the model because of their minor effects on volume and pressure. The relationships between pressure and flow rates are nonlinear, as are the characteristics of the fans and turboblowers. The model was therefore developed as a series of nonlinear algebraic equations, which were then solved at a variety of conditions on an Odra 1325 digital computer.

A prerequisite of the modeling effort was to define scaling factors that could be used to determine pressures and flows within the actual system. It was assumed that all flows are turbulent, and that the flow at a given point is related to pressure drop as follows:

$$h = R(V)^2$$

in which: h = pressure drop (mm H_2O)

V = gas flow (Nm^3/h)

R = scaling factor [(mm H_2O)$h^2/(Nm^3)^2$]

Two system configurations were selected for analysis, 1) operation of two converters and two ESP's, and 2) operation of one converter and two ESP's. Both cases assumed the operation of three lines at the acid plant. In the illustrations at the end of this appendix, Figure B-1 represents the acid plant system schematically, and Figures B-2 and B-3 represent the two cases of converter operation. The scaling factors for the points indicated on these figures were then calculated based upon actual pressure measurements in the system and the best possible estimates of gas flows. The calculated factors are presented in Table B-1.

The fan characteristics in the system can be expressed as a parabolic relationship, as indicated by the fan manufacturer's typical curve, and as shown in Figure B-4. The equation of the curve is:

$$h = a - bV - c(V^2)$$

The constants a, b, and c can be derived empirically from known values of h and V.

The following relationships define the coefficients from only two data points:

$$c = \frac{y_1 - y_m}{x_1 - x_m{}^2}$$

$$b = -2x_{mc}$$

$$a = y_m - x_m^2 c$$

The fan coefficients were first calculated for a speed of 1000 rpm. For both the first- and second-stage fans (1000 rpm, $\Delta P = 0$, $\Delta V = 0$), the coefficients were found to be: $a = 435$, $b = 1.5 \times 10^{-3}$, and $c = 0.0375 \times 10^{-6}$.

Changes in fan speed are accommodated by relationships which are linear with flow change and vary by the square of the compression change, as follows:

$$c = c_{odn}$$

$$b = \left(\frac{n}{n_{odn}}\right) \cdot b_{odn}$$

$$a = \left(\frac{n}{n_{odn}}\right) \cdot 2a_{odn}$$

A reduction in gas flow (ΔV) therefore results in the horizontal shifting of the fan curve shown in Figure B-5. The fan coefficients change is as follows:

$$c_v = c$$

$$b_v = b + 2c + \Delta V$$

$$a_v = a - b\Delta V - c\ (\Delta V)^2$$

Figure B-6 presents the changes resulting from a reduction in system pressure (ΔP). This vertical shifting of the fan curve is defined by the following relationships:

$$c_p = c$$

$$b_p = b$$

$$a_p = a - P$$

The turboblowers in the acid plant operate at a constant speed of 1000 rpm. The fan coefficients for the smaller size turboblowers were found to be: $a = 1826$, $b = -53.8 \times 10^{-3}$, and $c = 0.8594 \times 10^{-6}$.

A program developed by Eugeniusz Kosek for calculating flows and pressures in a system of square resistance was used to model the gas flows in the converter system. It had previously been used for ventilation systems in mines, as well as sanitary engineering applications. For the present exercise, a program named SUST was prepared; it was based on the original program and written in Algol. The program utilized the Cross algorithum, which relates pressure drops in a system according to the following equations:

$$\Delta h = R(V)^2 \qquad \operatorname{sgn}(V) - h_w$$

$$\Delta V = -\frac{1}{2} - \frac{\Delta h}{\alpha RV + \frac{1}{2}b + (cV)}$$

in which:

$h \rightarrow \Delta P$ = pressure at primary element

h_w = pressure drop across fan

R = scaling factor, $\alpha = 1$

The accuracy of the solution is ensured by specifying appropriate values for Σ_n and Σ_v. Because the discrepancies in pressure changes are less than Σ_n, the resulting flow changes will all be less than Σ_v.

Preparation of the data for a program run consisted of the following steps. All branches in the system diagrams (Figures B-1, B-2, and B-3) were numbered in sequence and resistance tables were prepared. Each branch was identified with a positive or negative symbol to indicate a forward or reverse gas flow. The number of independent loops was then determined by the following equation:

$$L = M - N + 1$$

in which:

L = number of loops
M = number of branches (which may be less than MAX)
N = number of points

Although not required, the selection of independent loops accelerates program execution. The most rapid calculations are obtained by ordering in sequence the number of branches chosen beginning with those of lowest resistance. Those branches that close at least one loop are excluded. The branches in which the flows are in the direction of the loop are considered positive and the remaining branches are noted as negative. The loops selected for analysis are presented in Table B-2. Bypass of the acid plant turboblowers was not considered in either the single- or double-converter analyses.

To begin a program run, the following data are input to the computer:

- Number of branches
- Number of independent loops
- Number of fans
- Resistance table, with values in sequence finishing with zero; branch number sequencing is optional
- Fan table; fan number sequencing is optional
- Loops desired; sequencing is optional, but should finish with zero
- The scaling factor (α) and values Σ_n and Σ_v

A completed program run prints out the gas flow, pressure drop, and resistance in sequence for each branch. The program may be modified to allow consideration of changes in either resistances or fans, although changes in both require that the ending values be given. The program can print out the conflicting loops after each run, as well as all intermediate calculations. Preliminary balancing flow values can be introduced to speed up the calculations. They are given in the form of resistance changes after all data have been entered.

The calculated pressure and flow values were refined over a series of program runs using different resistances and fan operating parameters. After 10 iterations, values were obtained that compared favorably with actual measurements conducted on single- and two-converter systems. Figures B-7 through B-10 reproduce these results on schematic diagrams.

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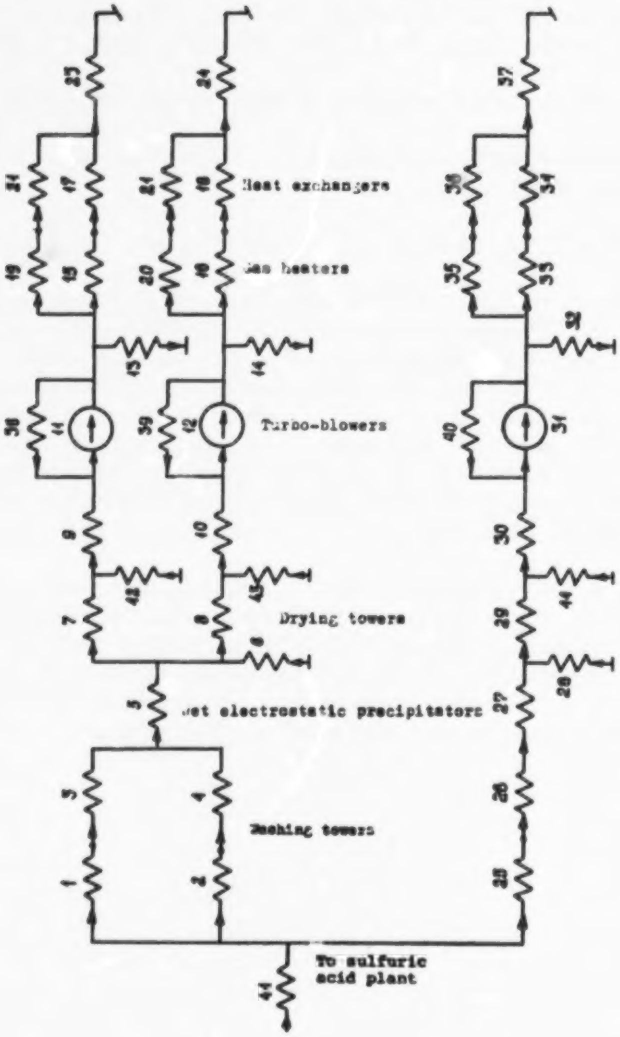
23
24
57
21
17
Heat exchangers
36
34
19
15
20
16
Gas heaters
35
33
32
13
14
38
11
39
12
Turbo-blowers
40
31
9
10
30
43
45
44
7
8
Drying towers
29
6
28
5
Wet electrostatic precipitators
27
3
4
Washing towers
26
1
2
25
To sulfuric
acid plant
44

Figure B-1. Gas flow through the acid plant system.

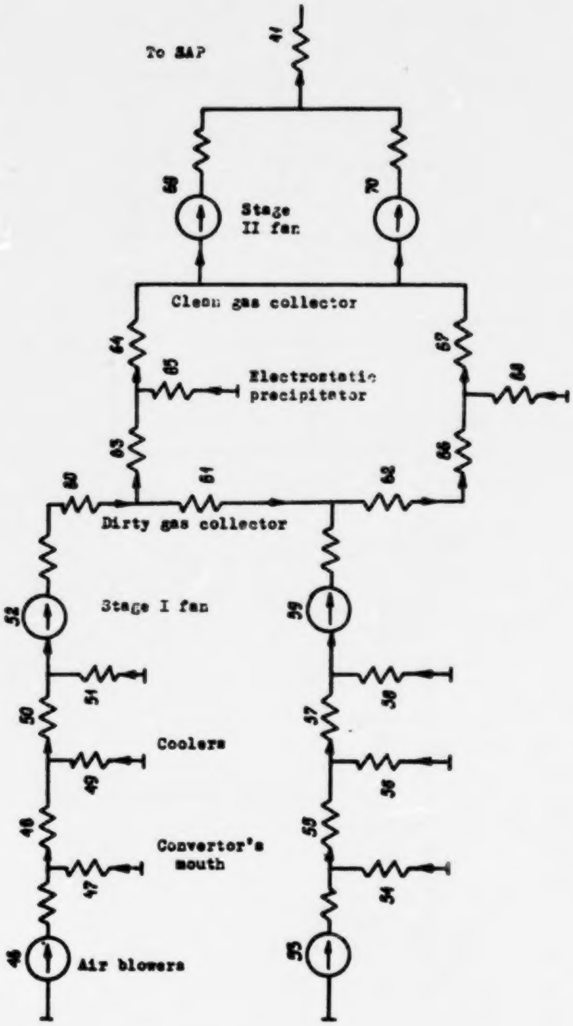
To SAP
41
69
70
Stage II fan
Clean gas collector
64
65
Electrostatic precipitator
67
68
63
66
60
61
62
Dirty gas collector
Stage I fan
52
59
51
58
50
57
Coolers
49
56
48
55
Convertor's mouth
47
54
46
53
Air blowers

Figure B-2. Gas flow through a two-converter system.

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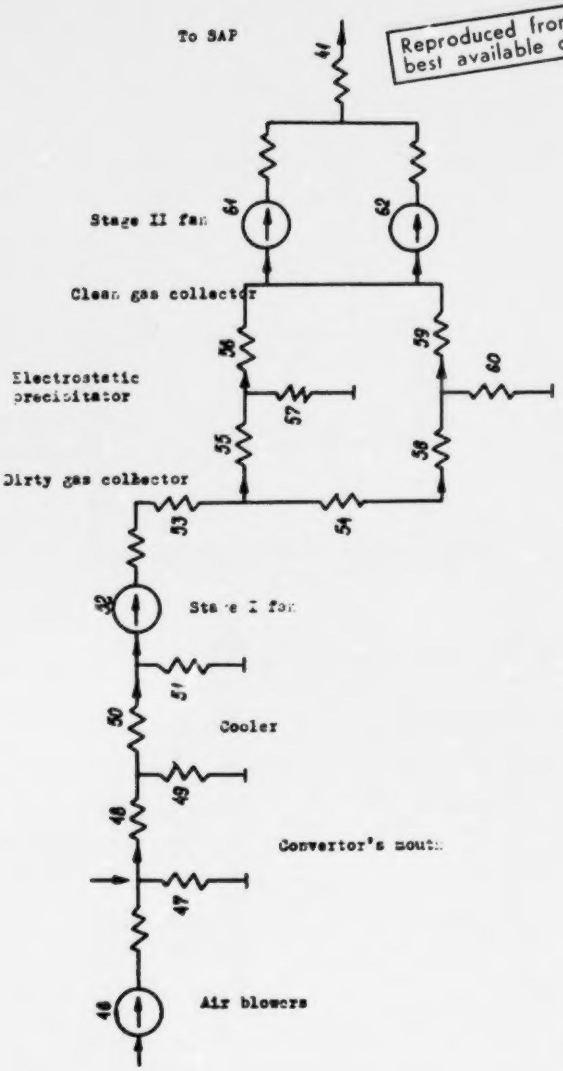
To SAP
41
Stage II fan
61
62
Clean gas collector
58
59
60
Electrostatic
precipitator
57
55
56
Dirty gas collector
53
54
52
Stage I fan
51
50
Cooler
49
48
Convertor's mouth
47
46
Air blowers

Figure B-3. Gas flow through a single-converter system.

TABLE B-1. CALCULATED SCALING FACTORS

Values of r for each numbered location:*

Sulfuric acid plant:

1) 0.0413
2) 0.05
3) 0.16
4) 0.2
5) 0.045
6) 5
7) 1.65
8) 2
9) 0.41
10) 0.5
11) 0
12) 0
13) 100
14) 100
15) 0.5
16) 0.5
17) 1.5
18) 1.5
19) 2
20) 2
21) 4
22) 4
23) 4
24) 4
25) 0.045
26) 0.175
27) 0.05
28) 5
29) 1.8
30) 0.45
31) 0
32) 100
33) 0.5
34) 1.5
35) 2
36) 4
37) 4
38) 20
39) 20
40) 20
41) 0.004
42) 20
43) 20
44) 20

Two converters:

46) 22.408
47) 0.8
48) 0.065
49) 8.8
50) 0.13
51) 20
52) 0.02
53) 24.3142
54) 1
55) 0.7
56) 10
57) 0.15
58) 30
59) 0.02
60) 0.008
61) 0.008
62) 0.008
63) 0.1
64) 0.08
65) 0.8
66) 0.1
67) 0.08
68) 0.8
69) 0.01
70) 0.01

One converter:

46) 22.408
47) 0.8
48) 0.065
49) 8.8
50) 0.13
51) 20
52) 0.02
53) 0.008
54) 0.008
55) 0.1
56) 0.08
57) 0.8
58) 0.1
59) 0.08
60) 0.8
61) 0.01
62) 0.01

---

* Locations refer to Figures B-1, B-2, and B-3. Scaling factors in table have been increased by 10^6 ($R = r \times 10^{-6}$).

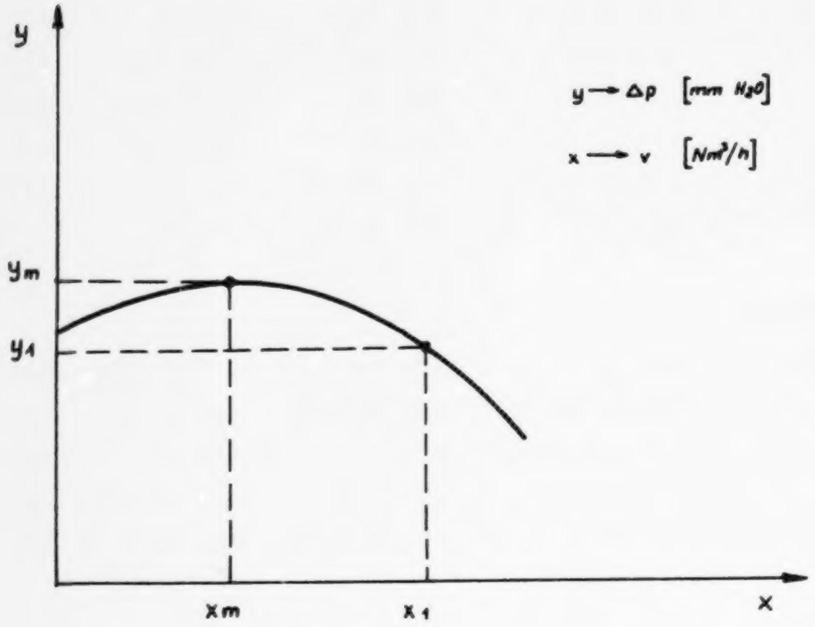

Figure B-4. Basic fan curve.

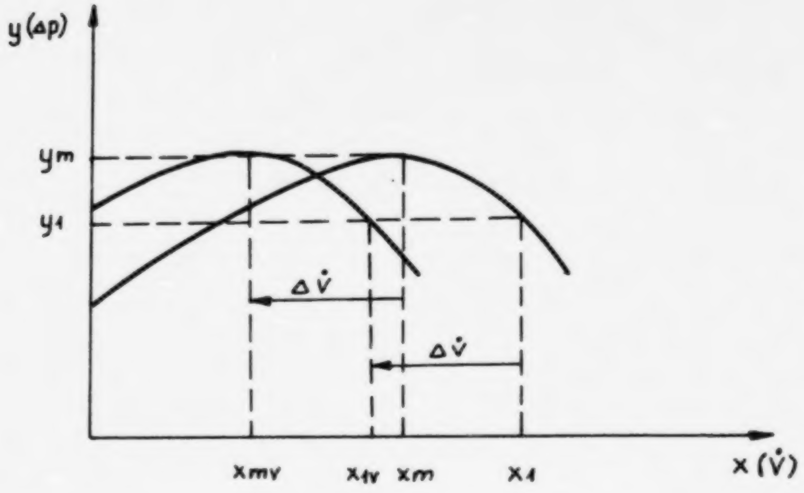
y(Δp)
ym
y1
ΔV̇
ΔV̇
xmv
x1v
xm
x1
x (V̇)

Figure B-5. Fan curves - reduced flow.

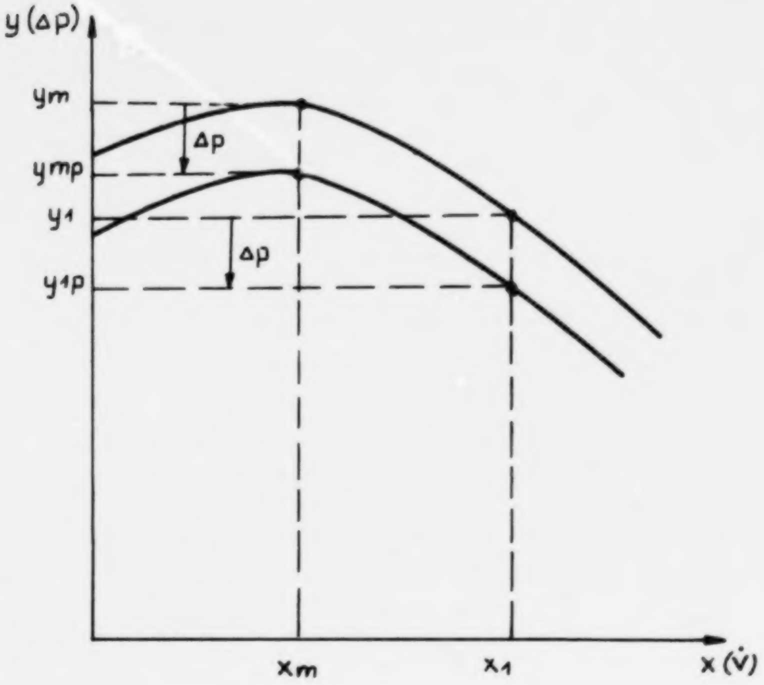
y (Δp)
ym
Δp
ymp
y1
Δp
y1p
xm
x1
x (V̇)

Figure B-6. Fan curves - reduced pressure.

TABLE B-2. LOOPS USED TO CALCULATE CONVERTER PRESSURE AND FLOW VALUES

Single-converter system:

MAX = 62, M = 58, N = 37, L = 22, $\alpha = 1$, $\Sigma_n = 0.01$, $\Sigma_v = n$

1) 46,-47
2) 47,48,-49
3) 47,48,50,-51
4) 55,56,-59,-58,-54
5) 47,48,50,52,53,54,58,59,-56,-57
6) 47,48,50,52,53,54,58,-60
7) 61,-62
8) 1,3,-4,2
9) 47,48,50,52,54,58,59,61,41,1,3,5,-6
10) 47,48,50,52,53,54,58,59,61,41,1,3,5,7,9,11,13
11) loop 10,5,8,10,12,14
12) 19,21,-17,-15
13) 20,22,-18,-16
14) loop 10,5,7,9,11,15,17,23
15) loop 10,5,8,10,12,16,18,24
16) loop 10,41,25,26,27,-28
17) loop 10,41,25,26,27,29,30,31,32
18) loop 17,31,33,34,37
19) 35,36,-34,-33
20) loop 10,5,7,-42
21) loop 10,5,8,-43
22) loop 10,41,25,26,27,29,-44

Two-converter system:

MAX = 70, M = 66, N = 41, L = 26, $\alpha = 1$, $\Sigma_n = 0.01$, $\Sigma_v = n$

1) 46,-47
2) 47,48,-49
3) 47,48,50,-51
4) 47,48,50,52,60,61,-59,-58
5) 47,48,50,52,60,61,-59,-57,-56
6) 47,48,50,52,60,61,-59,-57,-55,-54
7) 47,48,50,52,60,61,-59,-57,-55,-53
8) 63,64,-67,-66,-62,-61
9) 47,48,50,52,60,61,66,67,-64,-65
10) 47,48,50,52,60,61,62,66,-68
11) 69,-70
12) 1,3,-4,-2
13) 47,48,50,52,60,61,62,66,67,69,41,1,3,5,-6
14) loop 13,5,7,9,11,13
15) loop 13,5,8,10,12,14
16) 19,21,-17,-15
17) 20,22,-18,-16
18) loop 13,5,7,9,11,15,17,23
19) loop 13,5,8,10,12,16,18,24
20) loop 13,41,25,26,27,28
21) loop 20,27,29,30,31,32
22) 35,36,-34,-33
23) loop 21,31,33,34,37
24) loop 13,5,7,-42
25) loop 13,5,8,-43
26) loop 13,41,25,26,27,29,-44

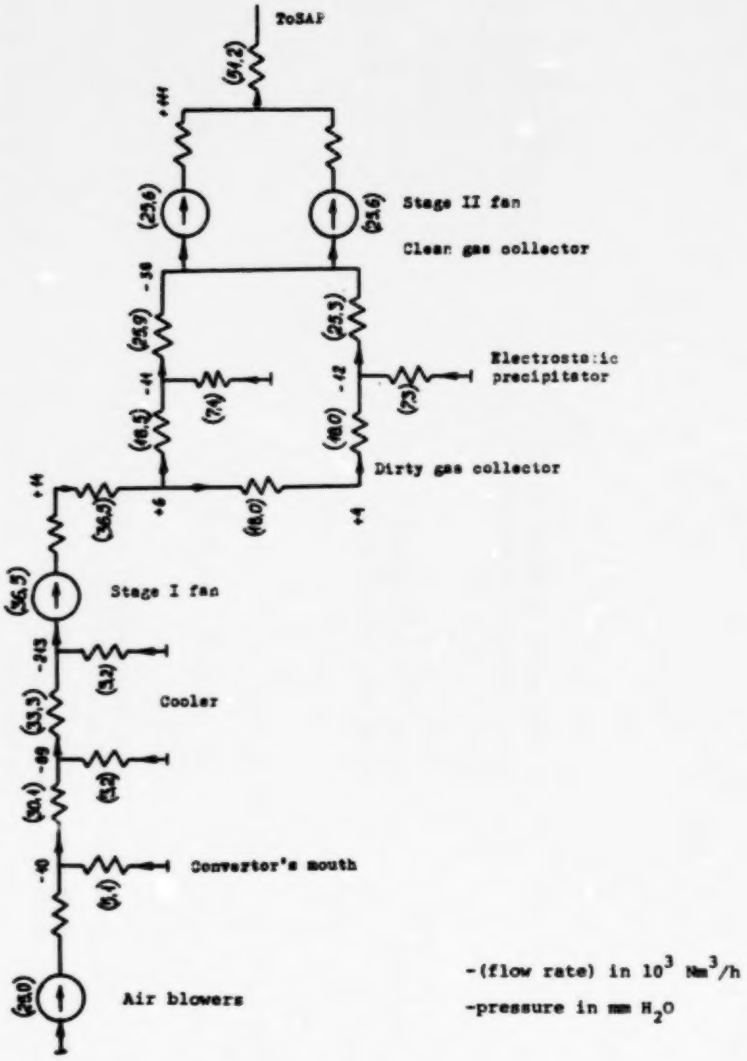
ToSAP
(54,2)
+444
(25,6)
(25,6)
Stage II fan
Clean gas collector
-38
(25,9)
(25,3)
-44
(7,4)
-42
(7,3)
Electrostatic precipitator
(18,5)
(18,0)
Dirty gas collector
+14
(36,5)
+6
(18,0)
+4
(36,5)
Stage I fan
-213
(3,2)
Cooler
(33,3)
-89
(3,2)
(30,1)
-40
(5,1)
Convertor's mouth
(28,0)
Air blowers
-(flow rate) in 10^3 Nm^3/h
-pressure in mm H_2O

Figure B-7. Gas flow and pressure through a single converter system.

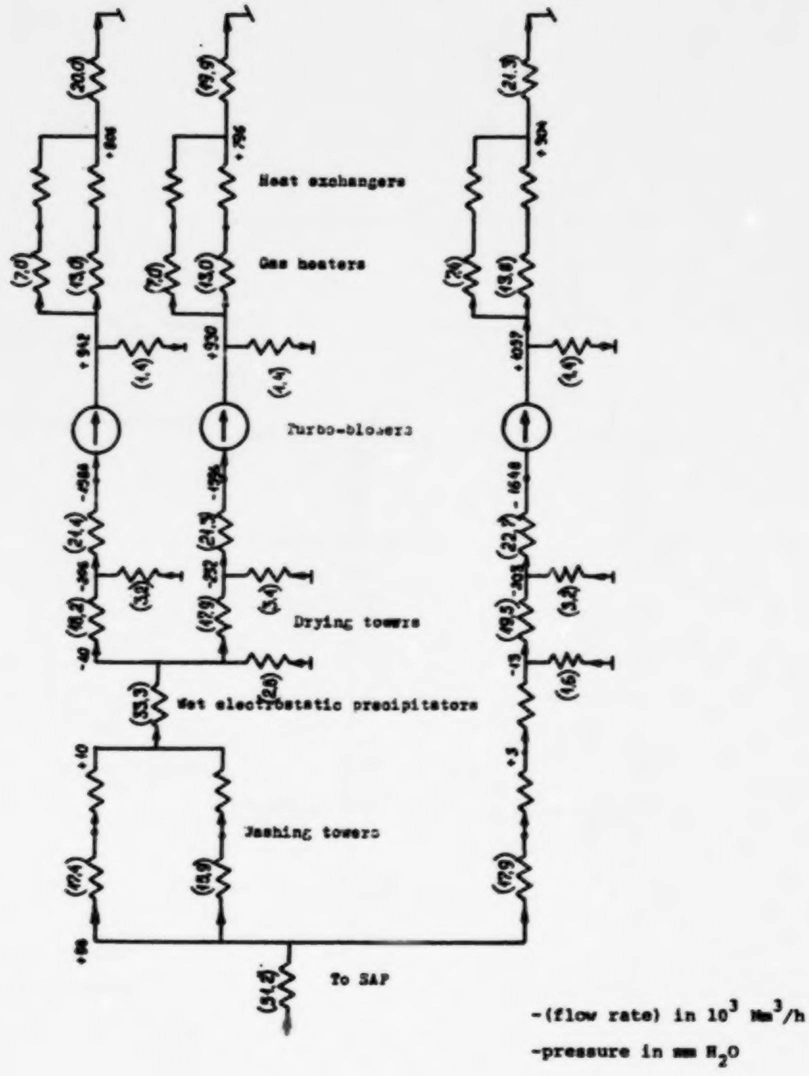
(20,0)
(19,9)
(21,3)
+808
+796
+304
Heat exchangers
(7,0)
(13,0)
(7,0)
(13,0)
(7,6)
(13,8)
Gas heaters
+942
+930
+1037
(4,4)
(4,4)
(4,9)
Turbo-blowers
-1588
-1596
-1648
(21,4)
(21,5)
(22,7)
-266
-252
-203
(3,2)
(3,4)
(3,2)
(18,2)
(17,9)
(19,5)
Drying towers
-40
-15
(2,6)
(4,6)
(33,3)
Wet electrostatic precipitators
+10
+3
Washing towers
(17,4)
(15,9)
(17,9)
+86
(51,2)
To SAP
-(flow rate) in 10^3 Nm^3/h
-pressure in mm H_2O

Figure B-8. Gas flow and pressure through a single converter system.

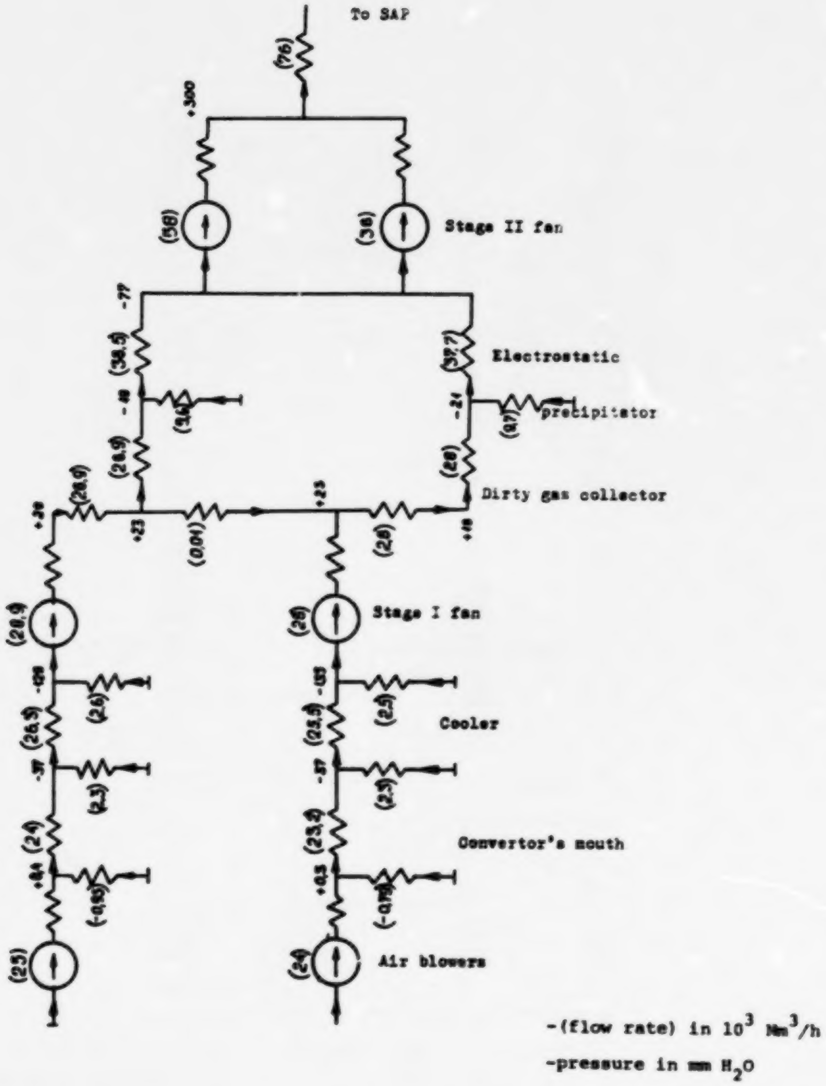
To SAP
(76)
+300
(58)
(38)
Stage II fan
-77
(38,5)
(37,7)
Electrostatic
precipitator
(9,6)
-24
(9,7)
(28,9)
(28)
Dirty gas collector
+30
(28,9)
+23
(0,04)
+25
(2,8)
+18
(28,9)
(28)
Stage I fan
(26,5)
(2,6)
(25,5)
(2,5)
Cooler
-57
(2,3)
-57
(2,5)
(24)
(23,2)
Convertor's mouth
+0,5
(25)
(24)
Air blowers
-(flow rate) in 10^3 Nm^3/h
-pressure in mm H$_2$O

Figure B-9. Gas flow and pressure through a two-converter system.

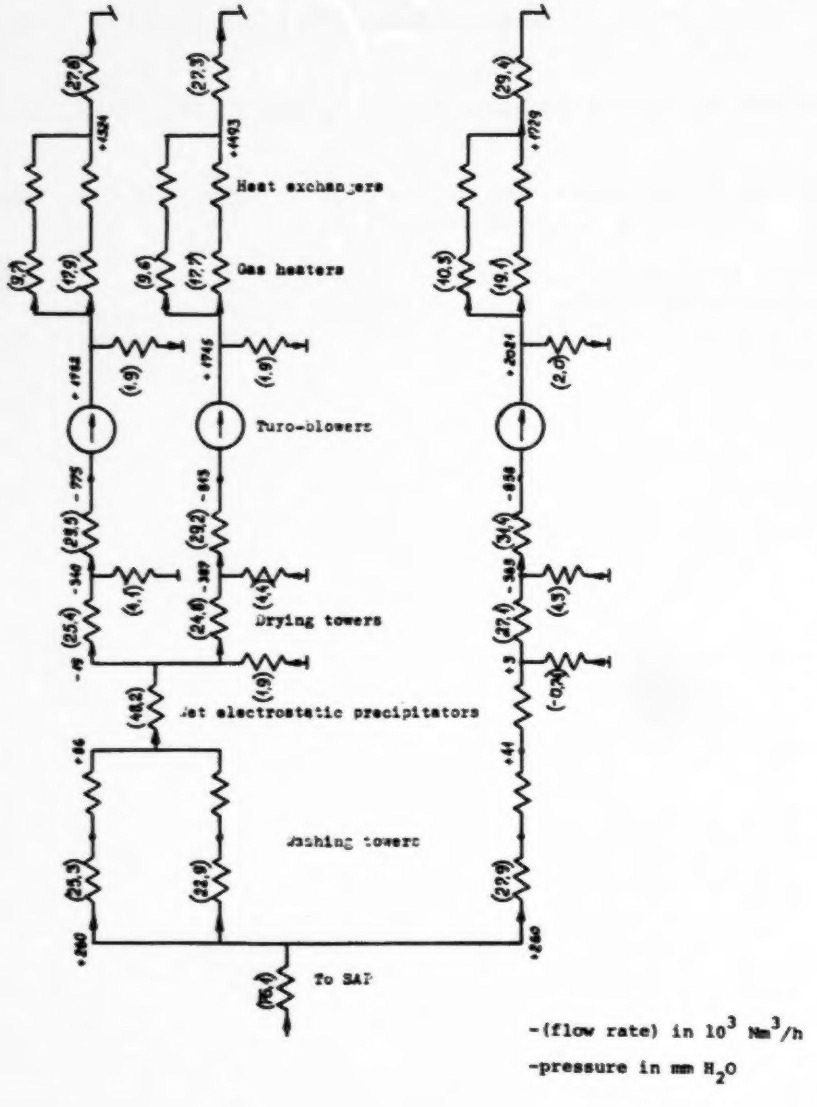
(27,8)
(27,3)
(29,4)
+1524
+1493
+1729
Heat exchangers
(9,7)
(17,9)
(9,6)
(17,7)
Gas heaters
(10,5)
(19,1)
+1762
(1,9)
+1745
(1,9)
+2021
(2,0)
Turo-blowers
-775
-845
-858
(29,5)
(29,2)
(31,4)
-340
-387
-365
(4,1)
(4,4)
(4,5)
(25,4)
(24,8)
(27,1)
Drying towers
+3
(1,9)
(-0,4)
(48,2)
Wet electrostatic precipitators
+86
+44
Washing towers
(25,3)
(22,9)
(27,9)
+260
+260
(76,1)
To SAP
-(flow rate) in 10³ Nm³/h
-pressure in mm H₂O

Figure B-10. Gas flow and pressure through a two-converter system

BIBLIOGRAPHY FOR APPENDIX B

Jerzykiewicz and Szczepkowicz. Algol 1204. PWN, Warszawa. 1972.

Nouvelles Techniques D'Etude et de Controle de la Ventilation Minere. C.E.C.A., Louvain. 1969.

Vang and Hartmann. Computer Solution of Three Dimensional Mine Ventilation Networks with Multiple Fans and Natural Ventilation. Intl. Journal Rock Mech. S.C.J. Pergamon Press. Vol. 4, 1967. pp. 129-154.

APPENDIX C

VERIFICATION TESTING

This is a summary of the work performed by Monsanto Research Corporation in verification testing for the Polish PL-480 project outlined in this report. The main objective of the work was to compare the particulate measurement method used for determining particulate loadings and concentrations in the effluent and process gas streams in Polish copper smelting processes to that used in U.S. smelters. Specifically, the program provides a comparison of the Polish method to the EPA Method 5 (EPA-5) as published in the Federal Register.

This project was completed by performing several subtasks:

(1) A presurvey visit was conducted to observe the smelting operation at the Polish Glogow smelter, to prepare for sampling that operation and to observe the Polish Particulate Sampling Method (PPSM).

(2) A sampling plan was prepared.

(3) Polish engineers and technicians were trained to use the EPA Method 5.

(4) Comparison testing was performed at the Glogow smelter using simultaneous sampling with PPSM and EPA-5.

(5) Analyses were performed on samples collected using both methods.

(6) Data were evaluated from the project and included in this final report.

A minor objective of this project was to attempt to estimate the sulfur dioxide (SO_2) emissions from the Hoboken style converter that is used at the Glogow smelter by making SO_2 concentration measurements of the ambient air at various points around the converter. This minor objective was not achieved due to a malfunction that occurred during the time of testing in the sulfuric acid (H_2SO_4) plant that controls the SO_2 emissions from the converter. Management at the Polish smelter site decided that sampling during this upset condition would be nonproductive because the concentrations during that time may not represent the true SO_2 levels present during normal operation.

SUMMARY AND DISCUSSION

In the testing program, particulate measurements were made at the inlet and outlet of an electrostatic precipitator (ESP) dust collector that controls the particulate emissions from a series of Hoboken converters. Simultaneous single-point runs were made using EPA Method 5 and PPSM trains with the probe inlet tips of each train located as close as possible to the same point in the gas streams.

Two runs (Nos. 1 and 2) were made on the inlet stream and seven runs (Nos. 3 through 9) were made on the outlet. Table C-1 presents the results of these runs and provides the particulate concentrations obtained using the Polish method and those obtained in both fractions of the EPA Method 5 train. The Polish method is a single fraction, in-stack filter procedure (as described later in this report) that is roughly comparable to the "front half" or "noncondensable" fraction of EPA Method 5. Therefore, a comparison factor has been calculated using the data from these two collections, and it is shown in Table C-1. Samples from Run No. 1 were not collected due to a broken cyclone in the EPA Method 5 train.

During the testing, it was quickly determined that the Method 5 equipment was not compatible with the conditions of the converter effluent gas. The high concentration of dust in the inlet line to the ESP caused rapid plugging of the filters and made short run times necessary. When longer run times could be used (at the outlet of the ESP), it was discovered that some of the materials used in constructing the EPA Method 5 train could not tolerate the strong acid conditions of the stack gas. It was estimated that the sulfuric acid dew point of the stack gas was in the range of 290° to 315°C. The silicone rubber material which had been used to seal the filter holder and hold the filter back-up frit was decomposed by the strong acid. Several attempts were made to seal the filter holder with other material until a suitable system was found. The new seal of steel and acid-resistant rubber materials could not be heated, and the filter was operated at just above ambient conditions. Table C-2 provides a brief description of the problems encountered with each EPA Method 5 run (and the measures used in attempts to correct the problems); Table C-3 briefly describes the possible effect of these problems and suggests the direction of the bias that may have occurred.

There was another problem that may have had a significant impact on the results of the comparison. Because the dew point of the stack was high, it was impossible to completely dessicate the EPA Method 5 filters and the particulate wash residues to

TABLE C-1. DUST CONCENTRATION OBTAINED USING BOTH TRAINS (g/Nm^3)[a]

Run No.	Date	Polish train, total concentration	EPA Method 5 train			Factor
			Front half	Back half	Total	EPA front half fraction / Polish fraction
2	10/17/78	7.46	35.57	0.48	36.05	4.78
3	10/17/78	5.49	15.87	0.09	15.96	2.89
4	10/18/78	0.05	14.88	1.22	16.10	298
5	10/19/78	0.90	19.72	NA[b]	19.72	21.9
6	10/20/78	0.88	10.31	2.55	12.87	11.7
7	10/20/78	2.23	19.98	1.83	21.81	8.96
8	10/21/78	0.71	19.69	0.18	19.87	27.7
9	10/21/78	0.23	11.74	1.28	13.02	51.04

[a] Normal cubic meter at 0°C for Polish data and 20°C for U.S. data, both at 760 mm Hg pressure.

[b] Sample lost due to an accident during analysis.

TABLE C-2. DESCRIPTION OF PROBLEMS ENCOUNTERED IN EPA METHOD 5 RUNS

Run number	Problem encountered
1 (Inlet)	Broken cyclone; sample contaminated with glass; did not save
2 (Inlet)	Filter plugged in 6 minutes; appeared to be a good run, except that filter was wet with H_2SO_4
3 (Outlet)	Filter plugged in 12 minutes; silicone rubber gasket appeared to be decomposing
4 (Outlet)	Filter plugged in 39 minutes; silicone rubber gasket destroyed (constructed gasket of Teflon)
5 (Outlet)	Filter frit was broken through the Teflon gasket; did not save back half of wash (constructed frit holder of steel and gaskets of acid-resistant rubber)
6 (Outlet)	Appeared to be a good run, except that filter was soaked with acid
7 (Outlet)	Filter backed off of frit and was bypassed
8 (Outlet)	Appeared to be good, except for acid on filter
9 (Outlet)	Appeared to be good, except for acid on filter

TABLE C-3. EXPECTED RESULTS FOR EPA METHOD 5

Run No.	Possible effect of problem and suggested direction of bias
2	Should be representative
3	May have silicone rubber on filter; high front half
4	Will have silicone rubber contamination; high front and back halves
5	Filter bypassed part of the time; may cause low front half
6	Should be representative, except for cold filter
7	Filter bypassed; low front half; high back half
8	Should be representative, except for cold filter
9	Should be representative, except for cold filter

dryness. As a result, all of the EPA Method 5 values are likely to be biased on the high side due to the mass of H_2SO_4 on the particulate.

The following conclusions can be drawn regarding the results of the tests:

(1) The EPA Method 5 can not be used to measure particulate concentrations in the Glogow converter effluent in its present configurations, due to the high acid dew point and the high particulate loading.

(2) As a result of the Conclusion No. 1, the Polish method and the EPA Method 5 can not be directly compared using this gas/particulate stream.

(3) In the comparisons that were made, the EPA Method 5 concentrations are likely to have been biased high due to an inability to completely dry H_2SO_4 from the particulate residues prior to weighing; this would result in a high comparison factor.

SAMPLING AND ANALYTICAL METHODS

The Polish Particulate Sampling Method

The Polish Particulate Sampling Method (PPSM) used at Polish copper smelters consists of elements and procedures similar to U.S. EPA Method 5 for particulate measurement. In the PPSM, preliminary tests are conducted to obtain velocity and temperature measurements of the duct to be sampled. A sample of the duct gases is then taken extractively through filter media and a flow metering system, so that the sample is extracted in a pseudo-isokinetic manner. The mass of the particulate collected on the filter media is then determined gravimetrically. A particulate concentration is determined by dividing the total mass of particulate collected by the total amount of gas sampled, and this is related to the total particulate loading in the duct by multiplying the concentration by total flow rate.

The velocity of the gases in the copper smelter ducts is determined by using a large P-type or standard pitot tube connected to an inclined manometer. The pitot tube, approximately 2.5 cm in outside diameter and 1.8 to 2.4 m long, has an approximately 30.5-cm long pitot sensing element facing into the gas stream. The stagnation impact area is a hole, approximately 11.5 cm in diameter, at the tip of the probe. The static openings, which are slots rather than circular holes, are located 8 to 10 pitot tube diameters from the stagnation impact hole. The static lines from the pitot tube are connected to a variable-angle mercury/alcohol inclined manometer. For our tests, the manometer at the copper smelter contained alcohol as a pressure sensing fluid and was set at a relatively low angle for sensitivity.

A scheme exists for laying out velocity traverse points; it involves dividing the duct into numerous equal-area quarter-circular regions, the number of which depends on disturbances in the proximity of the sampling point. The centroid of each of these areas is then sampled on two 90° traverses. This scheme, however, was not followed at the copper smelter; in fact, only one sampling traverse was used, and it consisted of 10 equally spaced traverse points across the approximately 1.2-meter diameter duct. Stack temperature was determined by inserting a metal-encased glass-mercury thermometer into the port to an immersion depth of about 20 to 25 cm. There were no provisions for determining the moisture content of the stream. No stack gas samples were taken for analysis to determine molecular weight; however, a device similar to our Orsat analyzer was available at the Institute, and it was indicated that the molecular weight of the stack gases had been determined in the past.

The particulate sampling equipment consists of an in-stack fiberglass plug filter, a probe, an umbilical line, a pressure manometer, a dry gas meter, a rotameter, and an air-operated vacuum aspirator. The filter consists of a steel cylinder with a perforated end cap in which fiberglass is packed. Another perforated end cap is placed on top of the fiberglass, and the whole device is desiccated and weighed. The filter fits into a probe nozzle head that is conical and faces into the stream to be sampled. The leading edge of the nozzle is sharp, and the opening is approximately 8 mm in diameter. This device is fastened to the end of the sampling probe, which consists of a short section of corrosion-resistant steel, a sweeping 90° bend, and a section approximately 3 meters long. An umbilical tube, which appeared to be made of reinforced neoprene rubber, is connected to the end of the probe and directs the gases down to the inlet of the dry gas meter. At the meter-inlet, a pressure tap is connected to a mercury U-tube manometer to indicate pressure on the inlet side of the meter. The dry gas meter is of the conventional bellows type. The gases exit the meter into a float meter (rotameter); from there they are connected by hose to an air-operated vacuum aspirator.

A sampling run is started by determining the initial tare weight of the filter cartridge in the laboratory. The filter is then assembled into the probe nozzle, and the remaining associated sampling equipment is connected. The probe is inserted through a packing gland in the stack wall and faces in the direction of flow at the first sampling point. The vacuum aspirator is started and the control valve opened until the calculated flow volume is indicated on the flow meter. Each traverse point is then sampled for a predetermined length of time, and the probe nozzle is moved to the next sampling point. At the completion of the run, the probe is removed from the stack and allowed to cool. The probe nozzle is then disassembled, and the filter element is removed to a desiccator. The desiccator is then moved to the laboratory where the increase in mass of the filter element is determined.

Several differences exist between the Polish method and the U.S. EPA Method 5. The EPA Method 5 uses real-time velocity measurements to determine sampling flow rate for isokinetic sampling at each individual sampling point. The PPSM uses a single sampling flow rate for all locations based on an average of the velocities taken at an earlier time. In the EPA Method 5, the sample stream is dried by removing moisture in a series of impinger condensers, and finally in a desiccant impinger, thus assuring that only dry sample gas reaches the dry gas meter. The PPSM uses no condenser due to the pressure drop inherent in such condensers and the limitation in the amount of vacuum allowed on the dry gas meter. This procedure can cause considerable condensation in the dry gas meter and the rotameter after a short period of operation. This pressure limitation on the dry gas meter also limits the PPSM method to relatively short sampling periods in highly loaded streams. Loading of the filter causes an increased pressure drop on the entire system, thus, run times must be shortened to accommodate the pressure increase. The EPA Method 5 suffers a similar pressure drop upon filter loading; however, the out-of-stack filter used in EPA Method 5 can be changed when the pressure increase becomes too high, and the meter of the EPA Method 5 train is not in the vacuum area between the filter and pump.

Another difference between the PPSM and the EPA Method 5 results from the temperature control of the filter media. In the EPA 5 Method, the filter located in an external oven can be controlled at a desired temperature. The PPSM has the filter located in the stack, and it must be operated at stack temperature. Therefore, particulates and aerosols that are volatile at stack temperatures may bypass the PPSM filter. This of course was a major source of trouble in the EPA Method 5 train in that sulfuric acid that bypassed the PPSM filter was condensed and collected on the EPA Method 5 filter. The final difference between the two methods stems from the operation of the PPSM dry test meter at various pressures, starting from the start of the run, under clean train conditions, to the end of the run when a vacuum of 80 mm of mercury could exist at the dry gas meter. The EPA Method 5 operates at a relatively constant meter pressure, differing from barometric pressure only by the ΔH pressure which is never more than a few inches of water above barometric pressure.

Figure C-1 shows a diagram of the Polish Particulate Sampling Method (PPSM) train, and Figure C-2 illustrates generally how the filter container appeared to have been constructed.

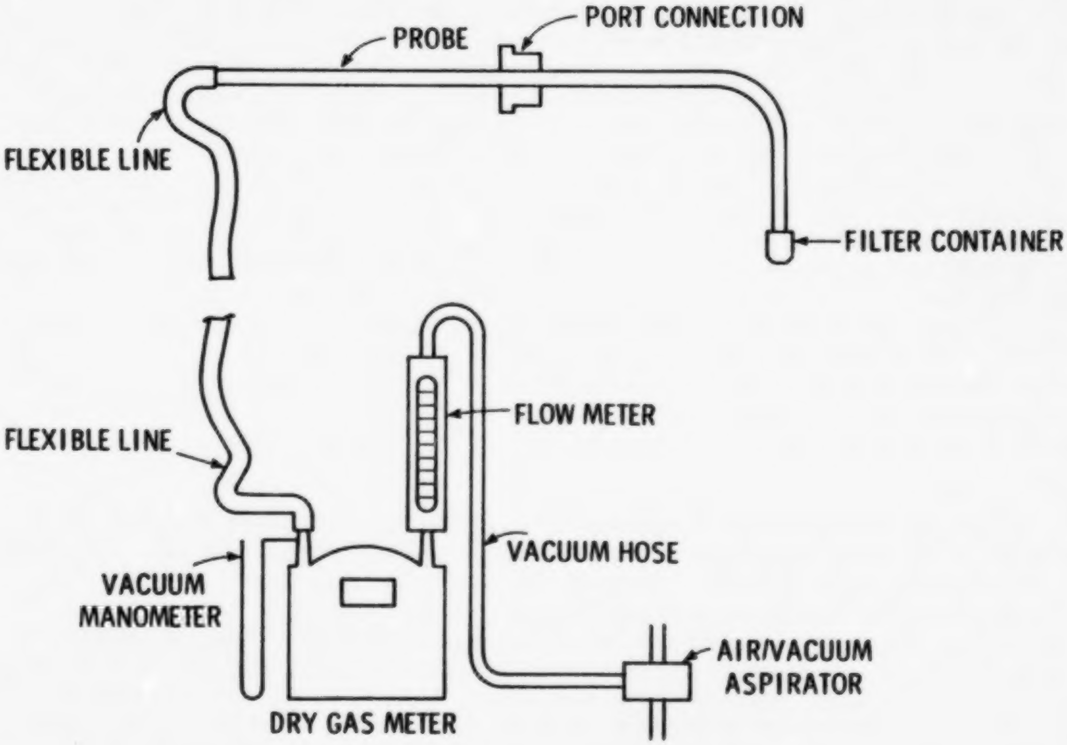
PROBE
PORT CONNECTION
FLEXIBLE LINE
FILTER CONTAINER
FLOW METER
FLEXIBLE LINE
VACUUM HOSE
VACUUM MANOMETER
AIR/VACUUM ASPIRATOR
DRY GAS METER

Figure C-1. Diagram of the Polish Particulate Sampling Method Train.

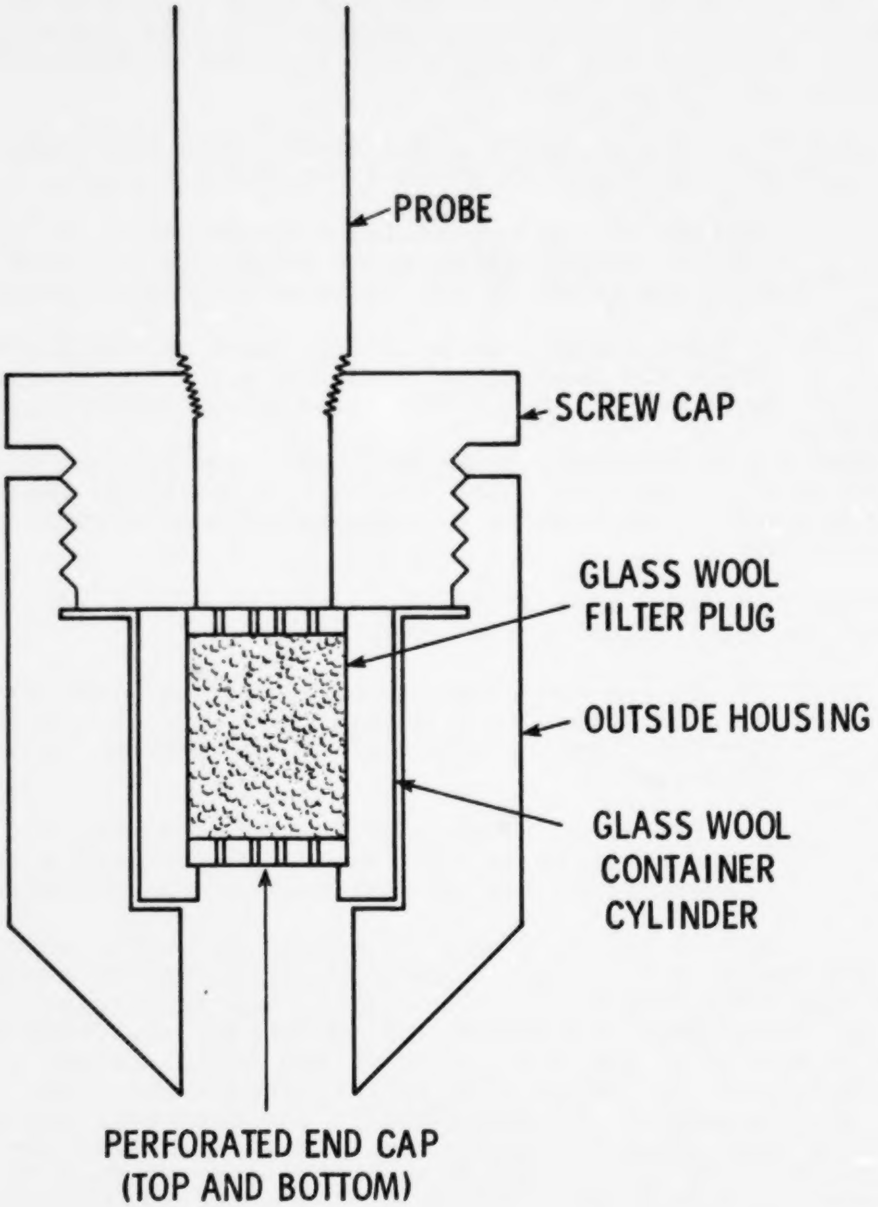
PROBE
SCREW CAP
GLASS WOOL FILTER PLUG
OUTSIDE HOUSING
GLASS WOOL CONTAINER CYLINDER
PERFORATED END CAP (TOP AND BOTTOM)

Figure C-2. PPSM filter holder.

The U.S. EPA Method 5 Sampling Method

The EPA Method 5 tests were conducted according to accepted practices and the instructions published in the Federal Register (42:160, Thursday, 18, August 1977) with the following three equipment modifications:

(1) A stainless-steel lined stack probe was used instead of the conventional glass-lined model.

(2) Instead of the conventional direct hook up, a flexible, heated, Teflon-lined probe extension was used to connect the probe to the cyclone and filter oven.

(3) A filter holder back-up frit made of steel and fritted glass was used along with two acid-resistant gaskets to hold the filter in the conventional glass holder.

The stainless-steel probe was used to eliminate breakage during shipment and use. The flexible probe connector was necessary due to the angles of approach encountered in entering the test port.

The following modifications to the procedure were necessary due to the conditions of the stack gases:

(1) Due to the high loading of particulate and H_2SO_4 mist, the EPA Method 5 runs were conducted at a single point for relatively short periods of time; no traversing was performed.

(2) The filter oven was operated at a temperature above ambient but below 38°C due to the inability of the gaskets to withstand heating above that temperature.

Cleanup and analysis were performed according to protocol. The EPA Method 5 runs were cleaned up according to accepted practices and published instructions. Dry particulate materials on the filter were desiccated and the weight was determined. The acetone washes of the front half of the train (probe, cyclone, flexible line, and front half filter) were evaporated, and the mass of the residue was determined. The back half inpinger contents were extracted with only chloroform (ether was not available), the extract was evaporated, and the mass of the residue was determined. The remaining water was evaporated at elevated temperatures, and the mass of its residue was determined. Back-half acetone washes were also evaporated, and the mass of the residue was determined.

Some of the associated preliminary and secondary testing normally used in conjunction with the Method 5 testing was not necessary. Preliminary moisture was assumed from data of previous work. Initial velocity and temperature traverses were made prior to the run's performance using the Polish pitot tube. A grab sample was taken and used to determine the molecular weight of the stack gas. Molecular weight analysis was performed by the Fyrite technique for determining oxygen and carbon dioxide content. Because the molecular weight of the carbon monoxide (if it existed in the stack gas) is identical to nitrogen, all remaining gases were assumed to have the molecular weight of nitrogen, except for the concentration of SO_2. SO_2 was present in a range that will affect the overall molecular weight of the stack gas; concentration values were acquired from plant personnel and the molecular weight of the gas was accounted for. Final moisture determination was accomplished using the water volumes collected in the Method 5 train. Isokinetic calculations were made in the field. All equipment was calibrated prior to shipment, and field checks for γ versus $\Delta H@$ were performed upon arrival in Poland. Because MRC's electric digital thermometer units are not accurate using the 50-cycle European current, a battery operated, calibrated, digital thermometer was used instead.

DESCRIPTION OF THE SAMPLING SITES

The effluent from the Hoboken converters is directed through ductwork to the outside of the converter building. At this point, the gases are at a relatively high temperature and are cooled by directing them through an atmospheric cooler consisting of several passes of ductwork exposed to the ambient air. From the heat exchanger/cooler, the gases are manifolded and separated into six ducts, one for each ESP unit. These ducts are A-shaped, extending approximately 7 m from the top of the cooler in an upward leg, making an approximately 45° downward bend for another 7 m and then entering the top of the ESP. The ESP inlet sample port is located in the downward duct just prior to entrance into the ESP. A single 5-cm port was originally located at this position on each ESP inlet; these ducts are approximately 120 cm in diameter.

The outlets of the ESP consist of horizontal sections of ductwork running from the side of the ESP building across the top of the control room building, and manifolding into a larger duct. These ducts are also 120 cm in diameter. Each duct originally had a sampling port that was a single, 5-cm diameter pipe coupling welded to the side of the duct at a distance of about four diameters from the ESP building. Access is gained by climbing to the top of the control building and erecting two sections of scaffolding to reach the duct.

The ESP is arranged so that any section can be isolated and repaired while the other sections are in operation. As many as three ESP sections can be down at any time, and the three remaining sections can handle the converter line load.

MODIFICATIONS FOR VERIFICATION TESTS

Instructions for modifying the ducts for comparison and verification testing were given to Polish engineers and referred primarily to the installation of additional test ports in several ESP inlet and outlet ducts. At the inlet, an additional 7.6-cm pipe port was located so that the sampling nozzle of the EPA Method 5 probe could be placed in the same plane and at approximately the same sampling point as the Polish probe. This port was located a distance upstream from the 5-cm port equal to the length of the sweeping bend section and short nozzle section of the PPSM probe. On the outlet duct of the ESP, a similar port was located just upstream from the current port and at the same distance from it, as was done on the inlet duct. An additional set of ports (two) were located two diameters further downstream of the 5-cm port. These consisted of a pair of 7.6-cm pipe ports located 90° apart and 45° from vertical on the bottom side of the duct. Sufficient scaffolding was erected to gain access to the new set of ports. Scaffolding was already in place at the old port location. The following additional modifications were required:

(1) An electrical extension cord was provided so that 220-volt, 50-cycle power would be available at the test locations. Originally, only 24-volt lighting power existed there.

(2) Sufficient scaffolding and equipment platforms were built.

(3) A means of raising and lowering the sampling equipment was provided.

Sketches of the sampling locations are shown in Figures C-3 and C-4. An illustration of the U.S. and Polish probe alignment is shown in Figure C-5.

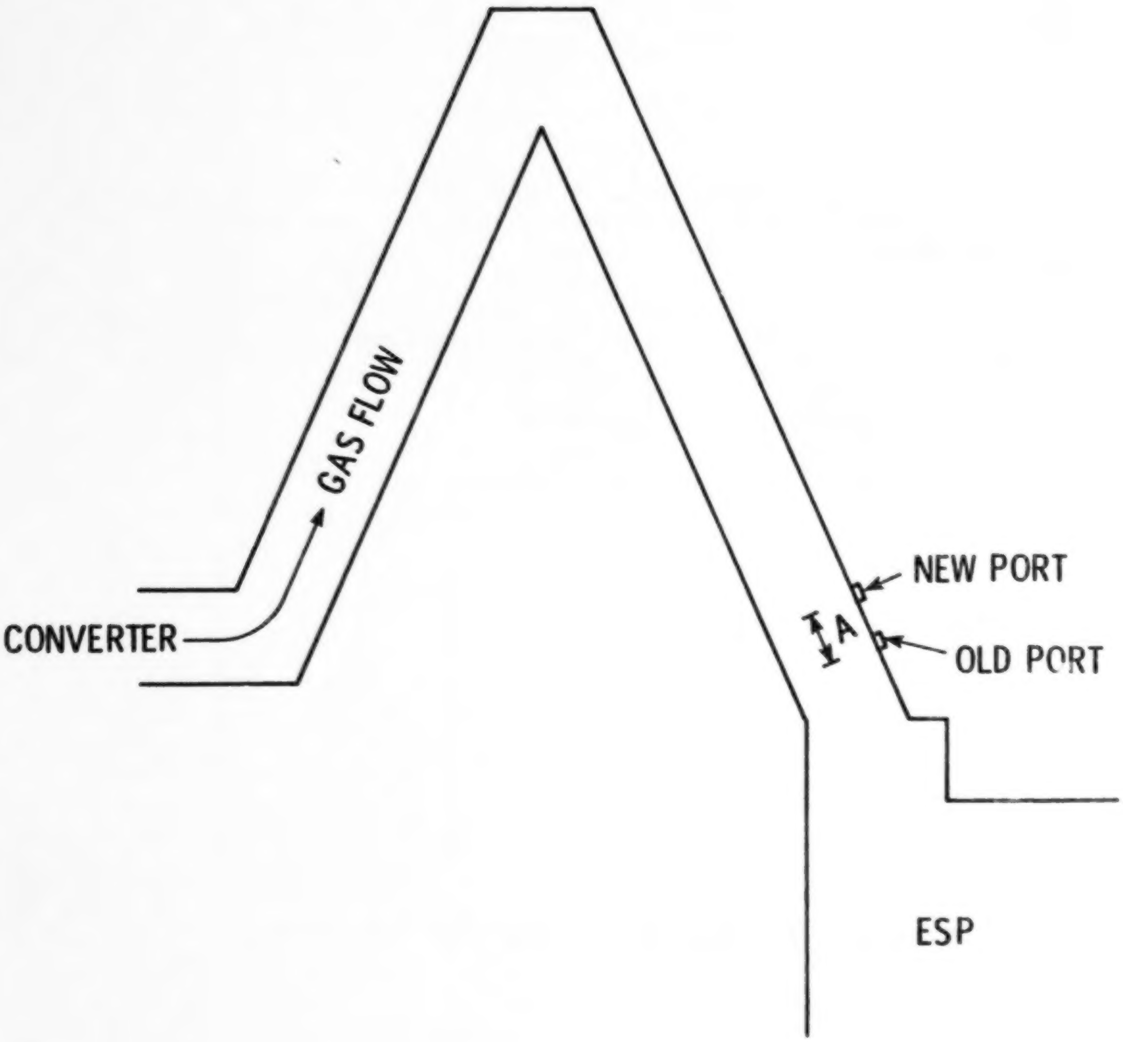
GAS FLOW
NEW PORT
CONVERTER
A
OLD PORT
ESP

Figure C-3. Sketch of sampling locations.

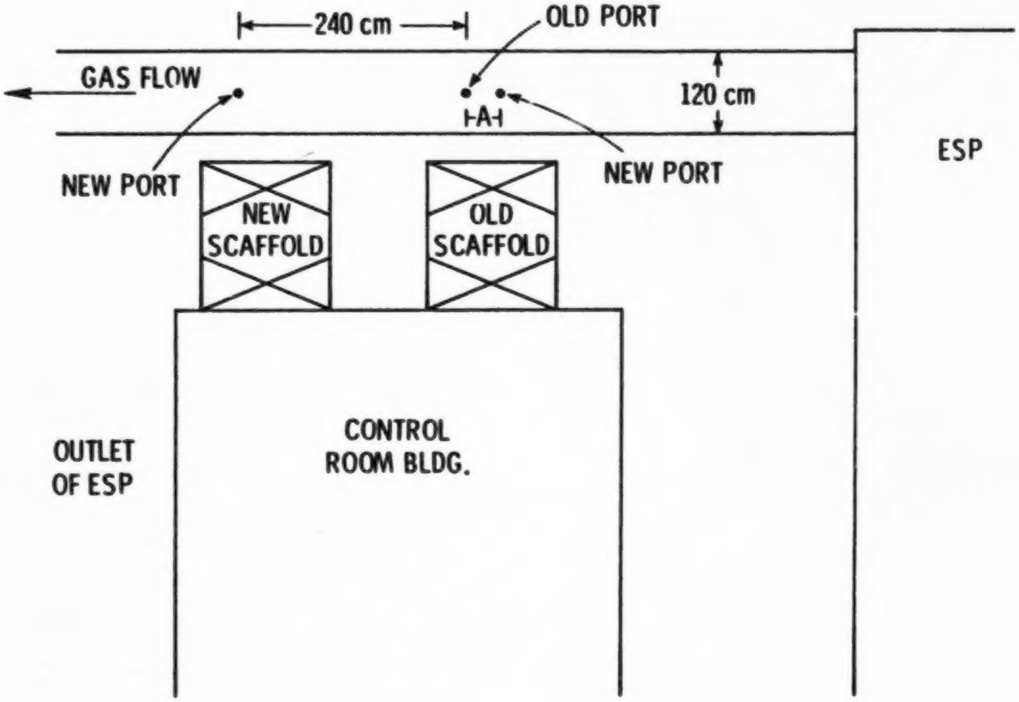
240 cm
OLD PORT
GAS FLOW
120 cm
A
NEW PORT
NEW PORT
ESP
NEW SCAFFOLD
OLD SCAFFOLD
CONTROL ROOM BLDG.
OUTLET OF ESP

Figure C-4. Sketch of sampling locations.

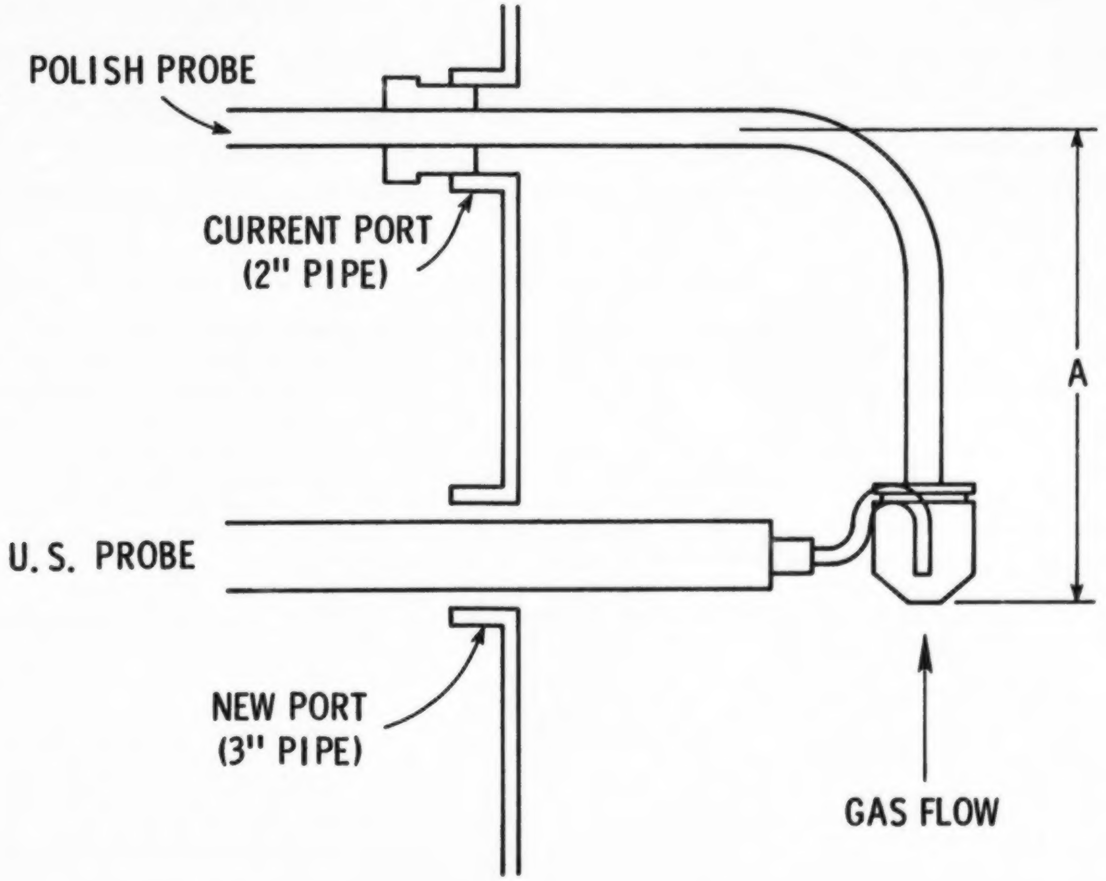
POLISH PROBE
CURRENT PORT (2" PIPE)
U. S. PROBE
NEW PORT (3" PIPE)
A
GAS FLOW

Figure C-5. Probe alignment.